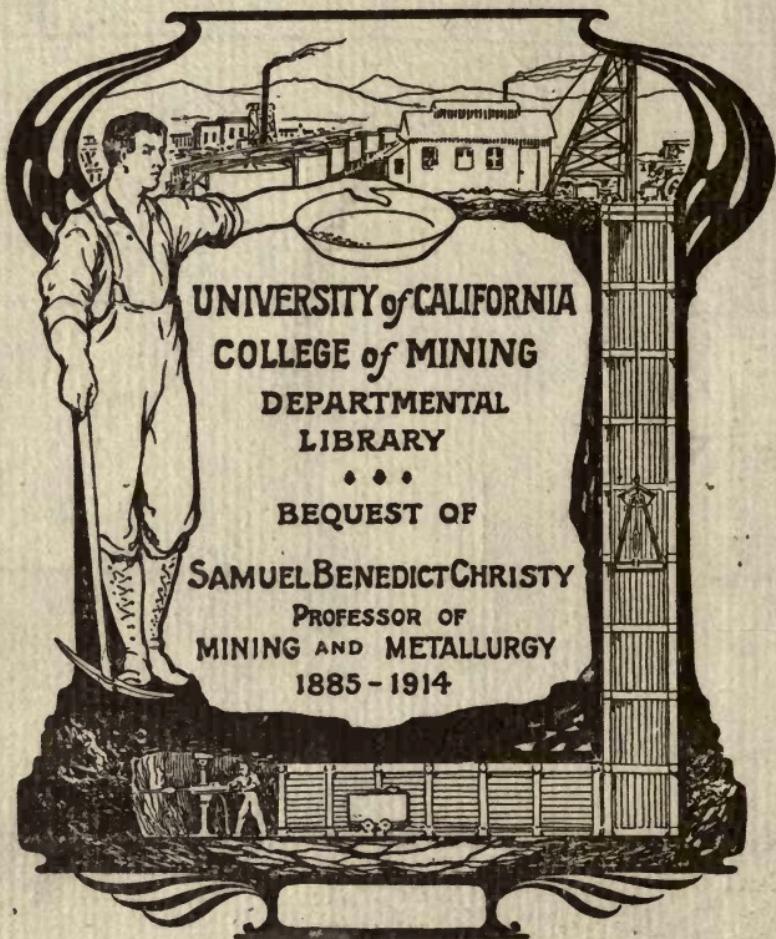


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CYANIDE PROCESSES.

BY

E. B. WILSON, E.M.
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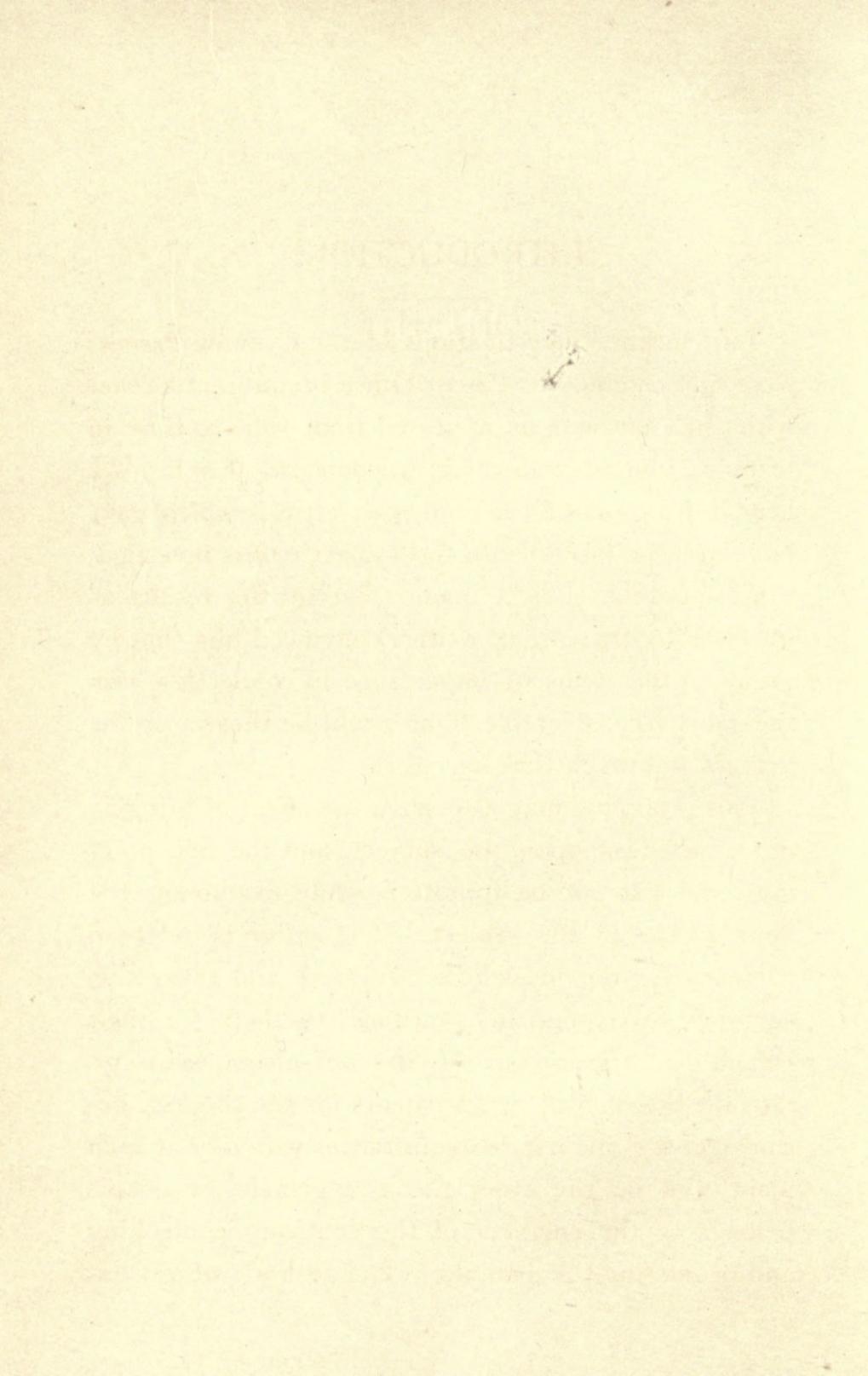
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INTRODUCTION.

THE author's investigations of the Cyanide Process were first confined to Patent Office literature, together with such extracts as appeared from time to time in technical journals and society periodicals. For his own benefit he grouped the numerous articles which gave but a passing insight into the subject. His investigations upon the subject made to verify the results as set forth by the various writers convinced him that by grouping the items of importance in connection with the subject considerable light could be thrown on the process and much time saved.

This grouping may also have the effect of bringing out new ideas upon the subject, and the one point mastered will not be forgotten while examining the next article on the subject. The writer is indebted to many for the ideas here advanced, and takes this occasion to express his gratitude to them for their valuable contributions. He has not attempted to go into the mechanical arrangements for the process, because on the one hand circumstances will vary at each plant, and on the other the arrangement of details belongs to the engineer of the company controlling and promoting the patents. The author's object has

been to confine the text to the process itself and such points as have special bearing on its success. While the rudiments of this method were known as early as 1806, it is only within the past few years that its practical value has been acknowledged, and even at the present time many prefer to use more complicated processes rather than change from a plan they are familiar with to one which is comparatively unfamiliar.

The writer has endeavored to be unbiassed in his remarks, but after reading and rereading some thirty different patent specifications with their claims, he fears stress enough has not been laid on defective points, and begs the reader's consideration if his opinions as set forth favor any one patentee more than another.

What he desires to bring out clearly are the points which he considers of value as he sees them, and to avoid those whose value was yet to be demonstrated. An examination of these numerous patents leads him to the conclusion long ago reached by some eminent authority that "there is nothing new under the sun."

For the benefit of those interested in mines he has dilated somewhat on stamp-mill work; for the benefit of the miner he has made remarks which he hopes will be of value; and lastly, for the mining engineer he has told all he knows of the subject at the present time.

THE AUTHOR.

CYANIDE PROCESSES.

CHAPTER I.

PRACTICALLY speaking, there are but two classes of ores which contain the precious metals: free-milling, or those which deliver up their gold contents to amalgamation, and refractory, which only do so in a measure or not at all.

The term “free-milling” is derived from the gold in the ores being in such a state that upon being crushed in a stamp-mill the particles of gold separate from the rock, free, to be then brought into contact with mercury, and from which they can only be freed by distillation. This treatment is termed amalgamation.

The term “refractory ores” is derived from the fact that all gold-bearing ores are not “free-milling,” and will not amalgamate their gold with mercury; in other words, obstinately refuse to yield up their gold to the easiest method of recovery.

Free-milling ores do not yield all their gold, and refractory ores will at times yield some of their gold to mercury, but the circumstances connected with

such results will not change the names adopted, as will be shown. As we are dealing with ores with reference to their adaptability to the cyanide treatment, and since all ores so treated must be stamped or crushed for this treatment, we will consider that a survey of stamp-mill work should precede the details of the process.

In all work uniformity of product is desirable, but particularly is this the case in metallurgical work, to obtain the most satisfactory results. Of the two methods of crushing or stamping in use, namely, the wet and the dry, the wet has given the better results both as to uniformity of the grains and speed of crushing.

When wet crushing is practised, water is allowed to flow into the mortar of the stamp-mill and flow out through a screen; sometimes two screens are used, one on each side of the mortar, giving a double discharge. The sized mesh used on these screens varies from sixty to one hundred for amalgamation, but for the cyanide process it has been in many instances possible to use 30-mesh screens and even very much coarser.

The size of the mesh of the screen determines the degree of fineness to which crushing is to be carried, as all fine particles which can pass the mesh are carried through by water, while all that cannot pass through remain in the mortar to be pounded until they can.

With dry crushing two discharge-screens can be employed to give more uniform results and faster pulverization.

When crushing dry, the stamps falling on the ore pulverizes it; there being no water to move it away, the dry powder accumulates in the battery mortar and under the stamps, thus retarding the process and very often forming a cushion for the stamp, not allowing it to crush particles uniformly.

The various machines so far invented have not proved satisfactory for dry crushing; and while with new rolls a very uniform product can be obtained, it is an endless repetition of screening and returning to the rolls again.

The best dry crushing so far witnessed by the writer is accomplished by Krupp's pulverizer; but even this is very slow, still there is no elevating and screening.

This field for inventive genius is open. There is needed very badly a good dry crusher which will crush uniformly, speedily, not clog with damp ores, not break with hard ores, will crush both hard and soft rock equally well, will not round the particles, but leave rough surfaces as do stamps, and will not be too costly for repairs. Any machine that can take up and meet all these requirements will take the place of stamp-mills. One of the obstacles to be overcome in crushing wet for the cyanide process is the amount

of water used necessary to do the work of uniform crushing. This water has been in some instances returned from settling-tanks back into the mortar to be used again and again. While there are no objections to the use of this water, there are to pumping it. It is understood that with wet crushing water enough must be used to allow the sludge to flow through the screens. It is often to the advantage of the cyanide process that coarse crushing is allowable, both as to amount of water used and time saved in crushing; still even this advantage of time is more than counterbalanced by time lost in treatment of ore by the cyanide solution when coarse crushing is practised. For if we are able to save time in crushing, and subsequent drainage of water and liquor, we must allow the cyanide solution to remain longer in contact with ore being treated.

On the other hand, when fine crushing is employed, as it must be at times, the water drains slowly from the settling-tanks, leaving the fine ore very firmly packed, and sometimes so firmly packed, if clayey or slimy, as to almost allow of no filtration, even when suction and other devices are employed to assist gravity. The average speed of drainage is about 12 inches per hour, which can be increased mechanically; but even in this case we shall still have the packing to contend with, which will interfere with cyanide leachings.

The character of the ore will determine the degree of fineness in crushing.

An open porous ore might be crushed to the size of a pea, while a compact ore with metal disseminated through it in grains may need 60-mesh or finer.

If the metal is in porous or loose ore in form of seams or streaks, coarse crushing will answer.

It is often necessary with refractory ores to use a preliminary treatment of lime to neutralize the acids which usually are found in partially oxidized sulphur pyritous ores, and are more or less dissoluble in water. Attempts have been made to neutralize these ores in the process of stamping by adding lime to the stamp-water. The objection to this proceeding is that a great degree of uncertainty must prevail as to the amount of lime required; and moreover, as the lime must be leached out of the ore by water later on, not much is to be gained, but considerable annoyance will be caused if too much lime finds its way into the ore.

Mr. A. B. Paul used a cyanide solution in the stamp-battery, the object being to take up gold as it was liberated from the ore, thus hastening the operation and lessening the amount of water waste.

There are several objections to this practice which make it unadvisable, and which summed up are:

That loss of cyanide takes place; no coarse gold can

be taken up; and that its action is of no great assistance to the mercury to be used in such instances.

Also the amount of gold recovered by this method would be so small that the customary leaching must follow; hence no time is gained. Again, this system would not be feasible if sulphurets were being treated.

The résumé of stamp-mill work for our investigations amounts to this:

That the more uniform the particles are crushed the more complete will be the extraction results with cyanide;

That the degree of fineness required will depend upon the character of the ore and the method of the distribution of metal in the ore;

That porous ores require less fineness than compact ores;

That dry crushing does not give as uniform particles as wet crushing;

That the use of lime or cyanide solution in the batteries is not to be commended;

That where particles of free gold exist in any appreciable size mercury can be used in the stamp-mortar to facilitate recovery when sulphides are not being stamped.

CHAPTER II.

AMALGAMATION.

REFRACTORY ORES—TAILINGS AND CONCENTRATES.

UNDER the heading of refractory ores come those parts of free-milling ores which cannot be delivered of their gold by mercury. With good stamp-mill practice, there will float away in the slimes or tailings a percentage of gold if wet stamping is followed, and the same percentage of loss occurs when dry crushing is practised. The cause of this loss will be stated later on. Ores containing the precious metals, free-milling at the surface very often become refractory when they reach a depth where the elements cannot obtain access to them; this depth is usually water-level. The cyanide process is particularly adapted to this class of ores.

To obtain the gold from free-milling ores mercury is placed in the mortar of the stamp-mill batteries, and amalgamated copper plates placed so that the gold going over them with slimes may be captured by the mercury on the plates. This amalgamation, as it is called, will recover by good mill-work 60% of the gold in the ore.

The various reasons advanced why more gold is not recovered are that the particles are too fine, and that others are covered with a film of oxide, sulphide, or tarnish which prevents the mercury from coming in contact with the gold, thus preventing its uniting with it. These fine particles and tarnished gold are virtually then refractory. After they have left the stamps they are termed tailings. It has been customary to grind these tailings in a pan with mercury, or to concentrate them by vanners, bumping-tables, or other means. The cyanide process acts very readily and completely on most tailings, so that its introduction has been the means of recovering immense quantities of gold from tailings.

Those ores which carry sulphides, tellurides, arsenides, antimonides, and base-metal compounds are refractory. When containing considerable gold, if treated with mercury, they make the mercury "flour," or else sicken, so that both mercury and gold are lost.

Sickened mercury becomes so sluggish by admixture of base metals that it will not act upon gold. This mercury can be restored to its normal condition so that it is not a loss, but while it is sickened it is of no practical value for gold recovery.

Flouring of mercury is a much more serious affair. It is due to sulphides coating the mercury and preventing the globules of mercury from uniting when once separated. Continued working subdivides the mercury

more and more until it becomes a black slimy mass which will be found difficult to settle in water, and is carried off in the washings and lost.

The separation of this metallic slime from the heavier portions of the ore is almost an impossibility.

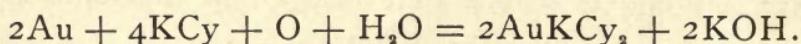
Various schemes have been proposed and various chemicals tried to obviate the difficulties encountered by flouring and sickening of the mercury, but we have heard of none which have proved entirely satisfactory to other than the inventor.

When fine or tarnished gold particles are met with, and the mercury retains its bright appearance and fluidity, amalgamation does not always take place. To force the gold to meet the mercury mechanical means, such as grinding with mercury, has been resorted to. If sulphides are present, this will cause the mercury to flour; if base metals are present, sickening of the amalgam will result. The proper method for treating such ores before the introduction of the cyanide process was concentration, or assortment for the smelters. As we are dealing merely with stamp-mill work only as it effects the cyanide process, we refer those who desire further information to Prof. T. A. Rickards' articles in the *Engineering and Mining Journal*, also to "California Gold Mill Practices" by Ed. B. Preston, which are the most replete with information of any in our language, at least that we have observed.

CHAPTER III.

FOUNDATION OF CYANIDE PROCESS.

THAT gold was soluble in cyanide of potassium solutions was known to Hagen in 1806. L. Elsner stated in 1844 that gold and silver could be dissolved in potassium cyanide without decomposition of water. He further stated that the dissolution of the metals is the consequence of the action of oxygen, which, absorbed from the air, decomposed part of the cyanides, thus forming a double salt auro-potassic cyanide, which has later been stated in the following equation, known as "Elsner's Equation":



The first scientific literature on the subject is by Prince Bagration in 1843. He concluded his paper with the remark that in the future cyanide of potassium must be enumerated among the solvents of gold. Faraday made use of a cyanide solution to produce thin films of gold in 1857. Ten years later J. H. Rae took out the first patent for applying cyanide to obtain gold from the ores direct. He was followed by Faucett in 1881; He in turn by Sanders in 1881;

then by J. W. Simpson in 1885; finally by MacArthur and Forrest in 1889. To these latter men is due the praise, not that they were the first discoverers, but because they pushed their inventions, and enabled the recovery of \$14,000,000 in five years by their process which otherwise would be irrecoverably lost.

It is of no special moment to us who discovered the process or whose process or patent is valid: that MacArthur people proved the way by practical demonstration is of more value to the world than who owns the patent. After the examination of some thirty or forty patent claims it appears to the writer that the patent offices are more at fault in granting claims than those who make the claims and obtain the patents.

Elsner's equation has caused such comment as to have led some to jump at conclusions too hastily. Even MacArthur stated that oxygen in the solution was not necessary for the dissolution of gold, but we believe he has repented. A pure solution of potassium cyanide will not dissolve gold to any great extent when immersed in it, but dissolves it readily when oxygen is present.

For example, if gold-leaf is placed on the surface of a cyanide solution, it will dissolve in a few minutes, and the stronger the solution the quicker it will dissolve; however, if submerged in the solution it dissolves but slowly, the strength of the solution effecting its rate of dissolution but slightly.

MacLaurin of New Zealand made some experiments which we quote from.

He used gold-leaf of uniform thickness, and his deductions were that oxygen was necessary for the solution of cyanide if it was to dissolve gold with any rapidity. A piece of gold-leaf placed in a stoppered bottle lost 0.18% of its weight in ninety-two hours.

Another piece placed in open bottle lost 9.1% of its weight in sixty-two hours. Still another piece placed in a bottle with oxygen lost 24.2% of its weight in ninety-six hours. The strength of the solutions was identical in every instance.

Experiments were made by the State Mining Bureau of California with dilute cyanide solution upon metallic gold.

With a 1 per cent solution it was found possible to dissolve such gold-leaf as is used by sign-painters in one hour. When dentists' foil was used, about six times thicker than painters' foil, it required forty-eight hours to dissolve.

Mr. J. B. Hanney verified Elsner's equation and MacLaurin's experiments in a practical way. His theory was that dilute cyanide solutions acted more rapidly on gold. He attributed this to the cyanide displacing oxygen by dissolving in the water, and that therefore gold could not dissolve until oxygen had been absorbed from the air. To carry out this theory he treated ores with dilute and then with strong solution.

He found that the rate of dissolution of gold increased with the strength of the solution, provided oxygen of the air could come in contact with the gold.

Mr. Hanney invented an electro-chemical apparatus which he used in his experiments. In direct opposition to this Mr. Miller has patented an apparatus to keep the air away from the solution.

From MacLaurin's experiments we deduce:

1. That oxygen is necessary for dissolving gold in a cyanide solution, and that it combines with the potassium of the potassium cyanide in the proportion required by Elsner's equation.

2. The rate at which gold is dissolved in a solution of potassium cyanide passes through its maximum in passing from dilute to concentrated solution, due to the fact that the solubility of oxygen in a cyanide solution decreases with the concentration. (See "General Information.")

The MacArthur-Forrest patent, May 14, 1889, consisted in subjecting the ores to a small quantity of cyanide solution without any other chemically active agent, but they state the process may be expedited by stirring.

Mr. J. C. Montgomerie added NaHO, then NaO₂, finally Na₂O₂, each oxide being the cause of a separate patent, and, as he says, giving equally good results.

Carl Moldenhauer claims that in order to extract gold it is necessary to simultaneously oxidize the solu-

tion. Instead of depending upon air and agitation he dissolves gold with cyanide of potassium in the presence of permanganate of potash. Later on he used ferricyanide of potassium in connection with cyanide of potassium to make the action more energetic, but states that ferricyanide of potassium will not dissolve gold.

THE SOLUTION.

The acknowledged practice for economical results is to use a weak solution of potassium cyanide. Mac-Arthur says regulate the quantity of cyanide so that its cyanogen will be in proportion to the gold and silver in the ore-charge, but in all cases to use sufficient water to keep the solution extremely dilute, because when dilute its preference is for gold and silver rather than the baser metals.

"We find it most advantageous to use a quantity of cyanide the cyanogen of which is equal in weight to from one to four parts for every thousand parts of ore, and we dissolve the cyanide in a quantity of water of about half the weight of the ore."

In other words, the solution should contain 8 parts of cyanogen to 1000 parts of water by weight. This proportion would give 20 pounds KCy to the 1000 pounds water, or 40 pounds to the ton of ore. This amount is not sufficient, except for the weak solution, at the commencement of the operation. The action

of cyanide upon gold is slow at first, but increases with time; hence by using a weak solution first to commence the operation the way is paved for the standard or strong solution, and at the same time the chances are lessened for the strong solution attacking the base metals present. There seems to be no definite rule regarding the amount of cyanide required for taking up one ounce of gold. Elsner's equation theoretically gives the amount as 1 part cyanogen for every 1.5 parts of gold to be dissolved. Practical results require 40 parts cyanogen to 1 part of gold, and from this down according to the care used and safeguards employed to economize its use.

Good results have been obtained with 0.05 per cent of cyanogen in the solution to the ton of ore.

The percentage of cyanide required will depend upon the nature of the ore and the methods used.

Losses must surely occur in treating ores containing acids or not thoroughly oxidized from acid salts.

They will also occur by Elsner's rule; again, by absorption of the vats; by coming in contact with carbonic acid; they will also occur in the zinc precipitation-boxes.

There may also some loss occur by coming in contact with iron

Mr. Paul says the proper per cent of the solution should be as testing will show, viz., the proper per cent for extracting the highest per cent of the pre-

cious metals, "say from one half per cent to one and one half per cent, then treating the ore for a given length of time with this solution."

Mr. Paul's rule is about the only rule we have to go by, and it must be the case from the nature of ores differing so widely in character.

Time is more an element for the successful operation of this process than a high percentage of cyanogen. As this depends upon the character of the ore, tests must be employed, as illustrated under heading of "Laboratory-work."

The action of the solution is to penetrate the ore, find the gold, and bring it out; if, therefore, coarse crushing can be used, time may be gained in crushing and rapidity of leaching, which will compensate for the extra time required for keeping the solution in contact with the ore.

Time may be saved by agitating the ore with the solution, either by mechanical stirrers or rotating barrels. These agitators are especially valuable for quick work where fine crushing is necessary or clayey slimes and tailings are to be dealt with. To facilitate matters filter-presses and mechanical suction of the liquid from the vats can be employed. In South Africa the vats and charges are so huge that mechanical stirrers would require considerable power, and as the vats stand by themselves, requiring no watching, no power is considered necessary. In other words, to

avoid the use of power they increase the size of the vats, saving time by the great charges treated.

Slimes or tailings from other processes can be readily treated by the cyanide method; in fact this makes it the most important all-around metallurgical factor yet put in practice, its use not being limited to free-milling or yet refractory ores, but adaptable to many and various phases of gold extraction, and when we say gold we mean silver as well, only using the term gold to avoid repetition of the words gold and silver.

Coarse gold requiring longer time for treatment by cyanide, it is better if possible to save what can be saved by amalgamation, and this latter would probably be followed because of the ease with which the bullion can be recovered from mercury.

When such coarse gold occurs in conjunction with fine gold or auriferous sulphurets, it is customary to crush wet, amalgamate in the stamp-mill mortar, pass the tailings into percolators or vats, which are provided with overflow-pipes to conduct the excess of water into a sump, from which it may be returned into the battery to be used again. In order to save a portion of the gold lost by amalgamation it has been customary to use concentrators, which save the metal grains containing gold and allow the sand to run off in the water. Tailings thus greatly reduced in bulk can be shipped to smelters or treated

again with mercury; the latter method will not prove very effective, however.

It is possible to treat these concentrates by cyanide, but, being generally richer in metal, their treatment requires longer time for the best results. As their quantity is limited, the size of the plant need not be as extensive as where the whole mass of tailings is to be treated. Agitation will hasten the operation with concentrates. Percolation with concentrates requires 20 days, because of the difficulty the solution encounters in passing through the coarse particles of ore. "Difficulty sometimes arises, owing to the crystalline form of iron pyrites and galena. These minerals crystallize in cubes, and in fluid arrange themselves face to face, so that a section of such a mass deposited from fluid would resemble a brick wall in structure." Dr. A. Scheidel suggests mixing coarse sand with the cubes to overcome the difficulty. (See "General Information.")

The results of the experiments made upon tailings containing sulphurets by the State Bureau of Mining of California are here given. A 1-per-cent solution of potassium cyanide was used for the experiments.

The experiments illustrate the effect of time and strength of the solution, while they also illustrate that the solution is not impaired in its action when sulphides only are present, both of which are important items in considering the scope of the process.

Treatment lasting 2 hrs. : tailings retained 35.29% gold.

3	"	"	"	31.37%	"
4	"	"	"	30.37%	"
6	"	"	"	25.49%	"
8	"	"	"	21.56%	"

Another portion from the same lot of sulphurets ground and passed through a 100-mesh screen (the former having been passed through 60-mesh sieve) were then treated six hours. There was left in the tailings after this treatment 17.64 per cent of gold, showing that fineness of the ore in this case materially aided digestion.

To ascertain whether dilute cyanide would extract all the gold in these sulphurets the ore was ground in an agate mortar to impalpable powder, and digested or 48 hours with three different solutions of 1 per cent cyanide, after which treatment the tailings were found to contain 9.8 per cent of gold.

Another lot of sulphurets from which free gold had been removed with great care was subjected to a 1-per-cent solution after passing 120-mesh screen.

Treatment lasting 2 hrs. : tailings retained 31.2% gold.

3	"	"	"	28.5%	"
4	"	"	"	15.6%	"
5	"	"	"	15.5%	"
8	"	"	"	10.4%	"

The deductions to be made by comparison with the

60-mesh screen experiments are that double the fineness lessens the time by one half in treatment of ore by cyanide solutions of the same strength.

We have so far arrived at the following conclusions regarding the necessity for oxygen and the strength of solutions:

1. That oxygen is necessary for dissolving gold in cyanide of potassium solution, and that it combines with the potassium of the potassium cyanide in the proportions stated in Elsner's equation.
2. That agitation shortens the process, as does the strength of the solution, but that what is gained in time is lost in power and cyanogen.
3. That weak solutions of cyanide of potassium are preferable to strong solutions, inasmuch as they are less prone to attack base metals, and are more economical in proportion to loss of cyanide during the operation.
4. That while theoretically one part of cyanide should dissolve one and one half parts of gold, practically the operation may require forty parts of cyanide to one part of gold.

CHAPTER IV.

CHEMISTRY OF THE OPERATION.

BEFORE the introduction of the cyanide process Plattner's method, with its improvements by Messrs. Thies, Rothwell, and others, was the only economical chemical process used on a commercial scale. The Russell process has found some advocates, but the methods of cyanide and chlorination have found more. The cloud which hovered over all chemical operations for the extraction of the precious metals, and which exists to-day, was due to frauds and un-practical schemes. Chemists have also made tests in their laboratories, and have induced parties to adopt their plans, which have proved to be valueless upon a commercial scale.

To obtain economical and practical methods simplicity must be one of the chief objects.

It is of no value that one can extract all the gold from an ore if the cost of extraction is more than the gold recovered.

While the cheapest piece of machinery is not always the best, the simplest and cheapest piece of mechanism which will do the work satisfactorily is the

best. The scope of a chemical process is another consideration; that process which will meet every requirement has not been found, but the cyanide process has approached nearer to the ideal than any we have knowledge of.

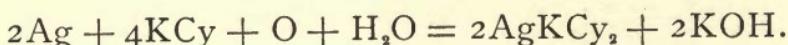
The chemical reaction of the cyanide process is such that after the chemist has determined the proportions of chemicals to be used in the treatment of any particular ore the entire work becomes mechanical.

The process may be adapted to wet or dry crushing; to coarse or fine ore, according to the texture of the ore; to tailings or concentrates from other treatment. It is not applicable to every ore, but to what ores it may be applied we can readily determine by analysis. Mr. Paul says that "if laboratory tests were satisfactory on 50 per cent of ore he would not hesitate to build a mill on that test, so uniform are the results to be obtained."

The chemical reaction expressed by Elsner's equation is for gold:



For silver:

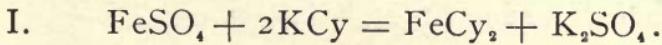


From the equations it is seen that double salts of gold and silver are formed with potassium cyanide, while caustic potash is liberated. It is yet an open

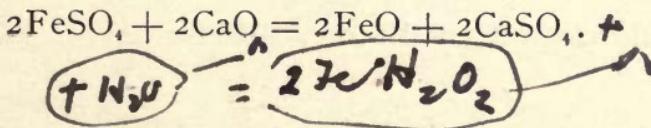
question whether gold is found otherwise than in a free state in ores, and it is given as a reason why gold unites so readily with cyanides that it is in a free state.

Silver is found more in combination with chemical compounds in gold ores, and hence is not recovered as readily as gold. Furthermore, the sulphides of zinc and other base metals generally accompanying gold and silver in refractory ores are not acted upon by cyanide solutions, at least so as to be noticeable.

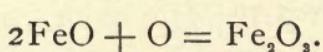
There are acid ores containing oxidized pyrites or acid earths which are very destructive to cyanide and must be treated before being introduced to the cyanide solution. Ores or tailings containing sulphurets subjected to the weather will oxidize, forming metallic sulphates on the earthy constituents. These are only partially soluble in water, so that while water washings will assist in removing them, other treatment should follow. The cyanide would partially be absorbed and partially decomposed by these substances if the ore were not freed from their influences, as illustrated in the following equations:



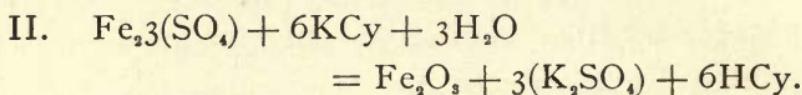
Were lime added before the potassium cyanide, we could expect the following reaction:



Ferrous oxide, while a powerful base, and neutralizing acid, absorbs oxygen readily, passing into ferric oxide, thus:

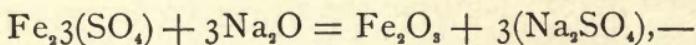


This being a very feeble base, it will have no further effect, and will undoubtedly be partially removed from the ore by the water wash necessary to carry off the calcium sulphate.⁽³⁾ MacArthur states that 1 part of lime is sufficient for 99 parts of ore for this treatment. We would advise, however, laboratory and litmus-paper tests for surety.



From this equation we see that ferric sulphate is converted into ferric oxide, and potassium sulphate with free hydrocyanic acid liberated as well.

The action of sulphates of alumina and magnesia is practically the same with cyanide, liberating free hydrocyanic acid, which is a loss. To neutralize these sulphates we give the ore a preliminary treatment as before, using lime or sodium oxide,—

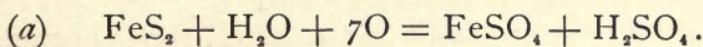


and then follow it with a water wash to remove the precipitants from the ore. While ferrous salts, solu-

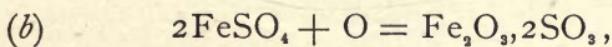
ble or insoluble, exist in the ore, the lime or soda will combine with the acid to deposit the ferric oxide. The ferric oxide, although a feeble base, would combine with cyanogen to a more or less extent, so that if it can be freed from the ore so much the better. Where sulphates of alkaline earths exist only, the oxidation may take place with the cyanide solution in contact, but this is not advisable where lime is present. The quantity of alkali presented for neutralizing acid ores must be determined by laboratory tests of the ore, as demonstrated under "Laboratory-work." The tests having determined the amount of alkali needed, the proper quantity is not added all at once, but little by little, so that its action will be uniform throughout the ore being treated.

Butters and Clennell advanced the following equations as possible reactions accompanying the action of cyanide on pyrites:

III. To illustrate the influence of oxygen on ferric sulphides or iron pyrites:

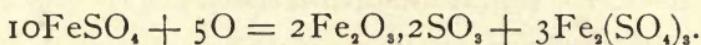


Iron sulphate and free sulphuric acid are formed by this action. Should the iron sulphate be further attacked by oxygen from the air, we could expect

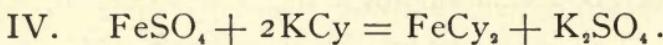


which is an insoluble basic sulphate (Wittstein).

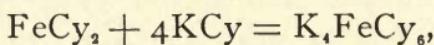
Berzelius gives the action on ferrous sulphate by the oxygen of the air as



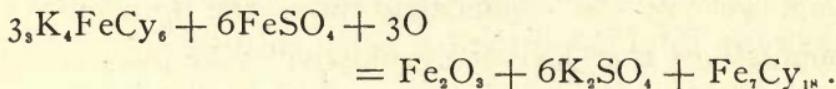
The sulphate being converted into insoluble basic sulphate and soluble ferric sulphate. The cyanide reaction on the ferrous salts is given as follows:



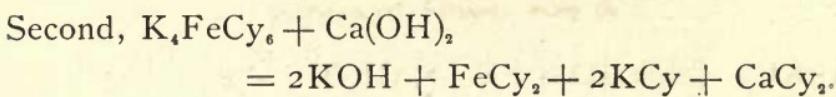
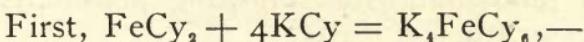
The reaction then continued results in



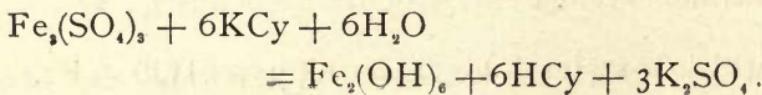
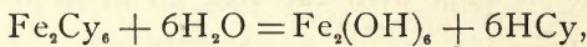
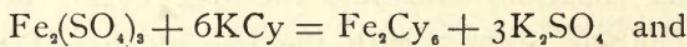
ultimately giving rise to



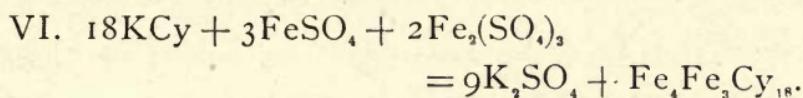
Should lime be present, the reaction would not be complete. Taking the double salt, as follows,—



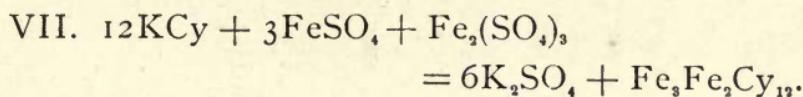
V. Ferric salts and cyanide give



A mixture of ferrous and ferric salts on addition of cyanide of potassium gives the well-known Prussian blue when ferric salt is in excess, thus:



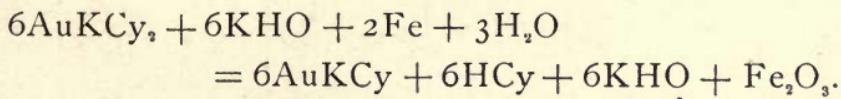
By mixture of the same salts with ferrous salt in excess we obtain Turnbulls blue:



The reactions between iron and potassium cyanide are very complicated and they are not fully understood, yet we can understand from the foregoing combinations that loss of cyanide will take place, so that the more safeguards used to prevent loss will prove economical and beneficial.

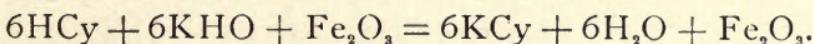
If iron be present in the solution of gold and cyanide of potassium, the following possibility may occur:

Does not occur.



hydro

In this equation, however, the free cyanic acid would combine at once with the free alkali to form potassium cyanide:



This reaction shows that if a metal be present it may be the means of regenerating potassium cyanide. We are inclined to believe that this action is one cause for so little loss of cyanide when the electro cyanide process is used.

The more recent practice is to mix with the cyanide solution sodium oxide to reduce any acid present or formed, and thus reduce the amount of cyanide consumed. J. C. Montgomerie uses NaOH , Na_2O , and Na_2O_2 with equally satisfactory results. His proportions of cyanide potassium to the ore to be treated are 3 lbs. of cyanide to the ounce of gold; $3/4$ by weight of sodium to the cyanide used. These are dissolved separately in a little water, then mixed together in water of sufficient quantity to treat the ore.

Novices should study the subject of sodium before mixing with water.

CHAPTER V.

LEACHING THE ORE.

THE original ideas of MacArthur-Forrest were to agitate the ore in the cyanide solution; when we say ore, tailings are included in the term. They suggested no form of vessel or mechanical means, but simply said that if the pulp were kept in motion by any convenient mechanical stirrer the operation would be hastened. Agitation should be continued until the gold and silver are nearly all dissolved, when the liquor containing the precious metals is to be drawn off for precipitation.

Rae's, Simpson's, and nearly every electrical cyanide system for the cyanide process is conducted on the lines of agitation. It is conceded that agitation lessens the time of extraction and expedites the process. It allows the oxygen of the atmosphere to act on the potassium, freeing the cyanogen so that it may unite with the gold; it is also probable that by agitation the cyanide can come into contact with the metals with greater ease: thus with tarnished particles of gold or gold covered with a sulphide film the slightest scratch would admit of the cyanide attacking it, and so

dissolving it in quicker time. Where ores are coarse and porous, agitation would assist the operation, but its great practical use will be found with ores hard and fine. Where very low-grade tailings are treated, or where fuel is expensive and unhandy to be obtained, or where very large vats and ore charges are used, mechanical agitation may prove more expensive than simple percolation.

Ores containing tellurides, arsenides, and sulphides will give better results if agitated than can be obtained by percolation. Barrels with arms, similar in construction to chlorinating-barrels, have been used; also shafts with radiating arms, and barrels simply rotated, for the purpose of agitation. The size of ore-charges treated is limited to the capacity of the barrels in one case and the size of the vats and motive power in the other.

If time be an element of importance, stronger solutions of cyanide may be employed with agitation, but as the loss is proportionally greater, it is the customary practice to use weaker solutions and economize on chemicals. With agitation-barrels it is necessary to introduce chemical oxidizing agents, otherwise no time will be gained, as has been illustrated by MacLaurin's experiments.

The percolating process is extensively employed in South Africa, where ores, concentrates, and tailings, are worked by it. This process is used as well in other

countries, and consists in soaking the ores to be treated in cyanide of potassium solutions. After soaking or leaching, as it is called, the solution is drawn off and another stronger solution run on.

As we have shown, the time required for running or draining these various leaching solutions from the ores depends upon the degree of fineness of the ore and its character for packing tight, which of course would hinder the rapidity of drainage.

Here it can be seen that uniformity of crushing is advantageous, and that the coarser the ore the quicker will be the drainage. The liquor is run into the vat through the bottom by a pipe connected with a vat holding the solutions of various strengths. The solution rises slowly upwards, percolating the ore in its upward movement, until it has covered the entire charge of ore in the vat. The object of this upward percolation against gravity is that by this means it permeates the mass evenly; It is run in slowly to avoid making channels in the ore, which would be disadvantageous to good downward percolation when drainage takes place. The idea is to make the whole quantity of liquor rise and fall through the ore evenly, and not up or down through channels.

When tailings, especially clayey tailings, are charged into vats, they are apt to remain lumpy. Cyanide solution will have difficulty in penetrating such masses, as channels will be formed and the liquor will naturally

go the easiest way through the ore. A mixture of clean sand with the ore will assist the percolation in such cases, and this proceeding may be absolutely necessary with clayey tailings which would pack without agitation and effectually prevent percolation. The process for cyanide treatment is conducted to this point as follows:

The ore in every instance is or has been crushed. It is then charged into a vat. If sulphates or other acid salts are present which would be injurious to potassium cyanide, the ore is allowed to stand in water, or water containing enough caustic alkali to neutralize the acid. This water is now drained off and wash-water added.

After the wash-water has drained off, the first solution of cyanide, containing from 0.02 to 0.08 per cent potassium cyanide, is allowed to percolate through the ore and stand some hours before draining.

The standard or strong solution is next percolated through the ore and drained off. After this another weak solution is percolated through the ore and drained off, and this last solution is followed by a water wash to take up any cyanide containing gold that remains in the ore.

The first weak solution is allowed to remain in contact with ore 12 hours.

The standard solution is allowed to remain in contact 12 hours, and has a strength from 0.5 to 1.5 per cent.

The final weak solution is run on after the ore has been exposed to the air 4 hours, and is allowed to remain about 12 hours. The final weak solution has a strength of from 0.02 to 0.04 per cent. The quantity of solutions used is one half a ton for each ton of ore.

The solutions of cyanide and gold are run from the vats into filter-boxes and from there into sumps, to be used again as weak solutions or else brought up to the standard of strong solution by addition of KCy.

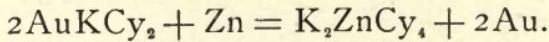
Having obtained the gold from the ore in potassium cyanide solution the next step is the recovery of the gold from the solution by precipitation. MacArthur's first patent suggests the evaporation of the solution to dryness, and fusing the resulting saline residue, or by treating the residue with sodium amalgam. Any one acquainted with evaporation on a small plan knows how tedious such methods are.

The treating of the residue with sodium amalgam would also be unsatisfactory, as the mercury would oxidize quickly and refuse to unite with the gold.

MacArthur was quick to see the crudeness of these methods, but by careful experiments with zinc finally ascertained fine filiform threads of zinc would precipitate the gold in metallic state. Thin zinc shavings are cut from disks of zinc turned in a lathe. The precipitation-boxes are filled with these zinc shavings, and are so connected together in a series that the

solution will go into the first box at the bottom, rise up through the zinc sponge until it reaches an orifice, when it flows into the next box of the series from the bottom up, and so on. The first solution in passing through the zinc filter does not precipitate the gold as fast as the strong solution, but its action on the zinc paves the way for quick precipitation of the gold in the strong solution. The action on zinc is slow, but increases in intensity after it has commenced. Zinc plates or granulated zinc will not answer for precipitation: the zinc must be in thin shavings. Pure zinc will not decompose pure potassium cyanide rapidly until some caustic is formed; but as soon as iron or gold is in contact with zinc and a cyanide solution, evolution of hydrogen commences at once.

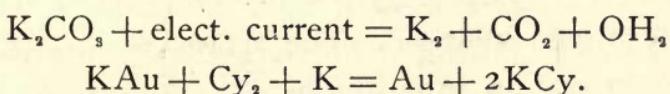
Hydrogen formed in the precipitation-box shows that electric action is going on between the metal and the solution, in which there is an exchange; thus,



This does not explain the H.

Alkali metals will precipitate gold from the cyanide solution. Mr. Malloy patented a process which, if successful, is superior in point of recovery to zinc precipitation, since the potassium cyanide is regenerated while gold is set free in mercury, from which it can be recovered readily compared with the recovery from zinc.

Malloy uses sodium or potassium amalgam, formed electrolytically from a solution of carbonate in contact with a bath of mercury. The alkali metal combines with cyanogen of the gold compound, forming an alkali salt of cyanogen, while the gold is instantly amalgamated. Thus,



If this method prove successful, it should add greatly to the cyanide process as an economical gold extractor, for it will save cyanide, and also recover finer bullion in a very much easier form to handle.

Dr. Johnston precipitates the gold and silver from the cyanide solution by pulverized charcoal, through which it is filtered. By one filter he recovers 25 per cent of the gold; by a series of filters he recovers 95 per cent. The gold and silver is recovered by smelting with fluxes after the charcoal has been disposed of by burning.

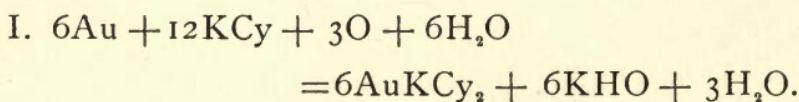
Attempts have been made to precipitate the gold by use of zinc and sodium amalgams, the object being to save cyanide and obtain the precious metals in the form of amalgam for recovery. Fairly good results have been obtained this way.

F. Rinder's process is to precipitate the precious metals by chloride of zinc. The gold is obtained as a grayish powder, by slowly dropping the chloride of

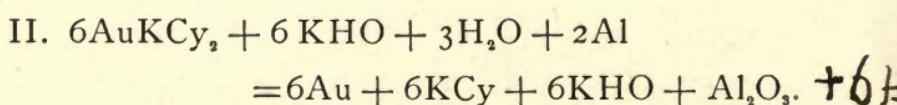
zinc into the solution. The precipitate is collected, dried, and finally melted into bullion. To obtain the gold and silver separately, the solution is treated first with sulphate of iron or sodium to separate the silver and precipitate it. The solution containing the gold is now filtered off from this precipitate, and the gold precipitated by chloride of zinc. The cyanide liquor can now be reinforced and used again.

Precipitation by electrical action is considered in the second part, under the proper heading.

Moldenhauer uses aluminum as a precipitant, which he claims does the work without entering into combination with cyanogen. He expresses his process as follows:



If to this cyanuret-of-gold solution, which contains free alkali, aluminum is added, the following reaction takes place:



The hydrocyanic acid combines at once with the caustic potash to form potassium cyanide. Thus the second member of equation II would be



and cyanide is regenerated.

In another instance he precipitates the gold and regenerates the cyanide by an alkaline earth, which may be added after precipitation by aluminum, provided a solution is used which contains a free acid and not a free alkali.

CHAPTER VI.

ZINC AS A PRECIPITANT.

THE objections to the use of zinc as a precipitant are such as to create considerable comment. It is considered too expensive by some, because of the loss of cyanide in the zinc-precipitating boxes, and because it requires so much zinc. Others consider it from a different standpoint, namely, that the recovery of bullion from its use is too coarse, and the recovery by refining too complicated, together with a certain loss of gold and silver due to refining those metals with zinc. The numerous methods suggested do not seem, however, to have driven the zinc-precipitating process to the wall, although we expect to see it accomplished in time.

“ Examination of precipitants from a cyanide mill, consisting of gold, silver, copper, calcium carbonate, and fine shreds of zinc, showed that copper was not in a metallic state, because it dissolved with effervescence in dilute HCl.” Mr. Eichbaum observed that zinc does not precipitate copper from a solution made with potassium cyanide 98 per cent purity; but

on adding iron a brisk evolution of gas took place, but no copper was precipitated. If an impure cyanide solution is used, copper is soon deposited upon the zinc, which is no doubt due to caustic or carbonate in the solution.

The solution, after gold has been deposited, contains zinc, yet the solution, after being used for months, does not cause inconvenience. By the addition of water the zinc could not saturate the solution, and the same is true of alkali carbonate, which, in the absence of lime, is continually forming. The white precipitate is the result of alkali on zinc, and the zinc potassic oxide on the double cyanide of zinc and potassium, as shown by the formula, and which is insoluble. This double cyanide of zinc and potassium being used over again in the percolating vats, forms, with the iron salts in the ore, ferrocyanide of zinc.

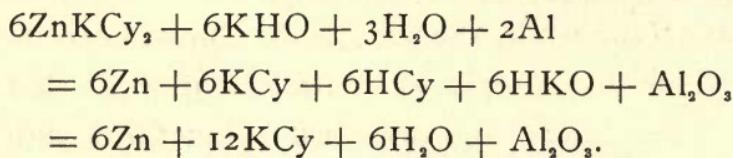
Buckland considers this the reason for constant removal of zinc from the solution with the residues.

This double cyanide of zinc and potassium formed during precipitation of gold is not available for dissolving gold in new operations, but it does not appear to be detrimental to the process when new cyanide solution is added to it. It does not precipitate gold dissolved by a new solution of cyanide.

Auro-potassium-cyanide seems to be a very stable compound, and not readily decomposed, as is evident when we see that, being once deposited on zinc,

the gold does not become redissolved by excess of cyanide so long as zinc is present.

We should think that if gold is readily deposited from potassium cyanide solution by aluminum, zinc could be as well, in which case the cyanide would be regenerated. Thus,



Zinc for precipitation must not contain arsenic, antimony, or lead, although a small percentage of the latter is not injurious. MacArthur's results from zinc tests were that fine zinc shavings answered the purpose better than zinc in any other form. Granulated zinc and zinc sheets produced unsatisfactory results, due possibly to the limited surface offered for attack, or possibly due to the slow action of cyanide on zinc in compact form, the same as its slow action on gold in grains.

We are inclined to this latter belief, and consider that it attacks the shavings because they offer numerous uneven surfaces or particles which the cyanide can attack readily. As before stated, while the action is slow at first, it becomes quite vigorous and rapid as it proceeds.

Zinc dust is objectionable, as it would pack and also

become mixed with other impurities, which would afford obstacles to refining the bullion.

Zinc amalgam does not afford surface enough for its weight, and the method of adding it to the solution in small pills, so to speak, is too tedious and faulty. The precipitation of the metals in the zinc-boxes takes place rapidly, and as the first series of boxes do most of the work, the zinc is consumed faster in them. This zinc is replaced by the zinc from the last boxes of the series, and the last boxes filled with fresh zinc. The object aimed at is to have the deposition of gold commence at once, which would not be the case with fresh zinc; besides, the few particles of gold adhering to the zinc in the last boxes are brought to the place where the greater part of the gold is precipitated.

According to theory, one pound of zinc should precipitate six pounds of gold. However, various losses occurring, such as generation of hydrogen, formation of potassium zinc cyanide, formation of zinc carbonate, and the loss which occurs from sifting the gold and ridding it of zinc, makes the total loss of zinc to gold precipitated sixteen to one practically. That the amount of zinc used will vary at different operations we can as readily expect as that different amounts of coal will be consumed under different boilers. One cubic foot of zinc shavings is said to be sufficient for the precipitation of gold from two tons of solu-

tion in twenty-four hours at some works. One cubic foot of zinc shavings weighs from three to six pounds, and exposes forty square feet of surface per pound. MacArthur says: "Improvements in detail to increase the surface and decrease the weight of the zinc have enabled him to obtain threads so light that one pound will occupy the space of two gallons." The zinc in this form is said to possess enormous chemical activity.

After passing through the zinc-boxes the solution should not contain 50 per cent per ton of gold or hardly a trace, since this gold will no doubt be lost by precipitation from the weak solutions.

CHAPTER VII.

TREATMENT OF BULLION.

HAVING precipitated the gold and silver from the cyanide solution, the next process is refining the bullion. The simplest method of recovering gold from the ore is by amalgamation with mercury. The mercury is then pressed through a cloth or chamois-skin, the gold amalgam remaining in the cloth or skin. This is then placed in a retort and the mercury distilled from the gold; the mercury is condensed and saved for future use. When the cyanide process is used, the operation is much more complicated. The zinc with the gold adhering is shaken in water, when the gold and zinc fall off into the water. The water is then filtered, and the fibrous particles of zinc may be collected in a sieve and shaken again to remove the gold. This last sifting removes the coarse zinc from the mass to be refined. The precipitate will contain gold, silver, and zinc, and if it comes from cleaning up of zinc filter-boxes other metals.

This precipitate is thoroughly dried and roasted slowly, care being taken that the flame does not come in contact with the mass. After being thoroughly

dried it may be refined by the “ calcining and roasting process” or recovered by the “ acid process.”

The roasting process is used where oxidation of base metals has not been complete. At Johnannsburgh, South Africa, the acid treatment is not employed, as it involves washing and filtration of the slimes, with loss of gold by formation of regulus in melting, if sulphates remain in the slimes, by fault of imperfect washing. “ The practice is to dry the slimes to dust nearly; then to thoroughly mix them with powdered nitre, the amount varying from 3 to 33 per cent of their weight, and the mass gently heated on a wrought-iron tray. The flames must not come in contact with the slimes, and the gases should be conducted up a flue away from the operator. By the use of nitre everything connected with the precipitate is refined. After this oxidization the roasted mass is placed in plumbago crucibles with the proper fluxes.”

When metallic oxides are present, the flux is: six parts roasted nitre slimes; four parts borax; two parts soda; one part sand. When only a small amount of metallic oxide is present, the charge may be three parts slimes, one part borax, two parts soda, one part sand. The function of the sand is to form a fusible slag with soda, and protect the pots from metallic oxides and potash formed by the reduction of the nitre. The slag resulting from the melting of these

slimes usually contains considerable gold; it is therefore crushed and sent to the smelters. It is sometimes customary to pan these slags, after crushing, for gold, sending the tailings to the smelter. Fluxes which give clean fluid slag are preferable in this as in any other refining.

For the sulphuric acid treatment wooden tubs are sufficient, the temperature created by the action of the acid on the zinc in dissolving it being sufficient to discard the application of artificial heat. After the zinc has dissolved the acid solution is decanted, and the bullion washed to remove all acid traces.

Should any zinc remain, its presence in melting will cause loss of gold by evaporation, as the zinc volatilizes by heat; it may be better, therefore, to treat the precipitate first with a weak solution of acid, then follow it by a stronger.

After washing the mass is dried at a low heat; when the moisture is driven off, the heat is increased to a dark red. After one hour's roasting all oxidation of the base metals which escaped removal by acid treatment should be complete, the mass presenting a brown-gray appearance. The roasting completed, the bullion is transferred to a wrought-iron box for cooling purposes. When cool, it is pulverized. Borax and soda are added to make a clear slag. Plumbago crucibles are well adapted for this treatment, and borax is put into them first; the mixture of borax and

roasted ore is put in from time to time as the mixture melts and sinks down. The melting goes on speedily and in the most satisfactory manner. After the whole quantity is charged the temperature is kept high for some time to give the small bullion globules time to collect. The contents of the crucible are then poured into a heated mould and allowed to cool. It is claimed that this acid treatment has advantages over the calcining treatment, being simpler and allowing the plumbago crucibles to be repeatedly used; also that the bullion is finer and by care can be recovered without chemical loss." (Dr. Scheidel.)

The bullion produced by the cyanide process must necessarily vary according to the grade of ore treated and care taken for its recovery and refining. While one party will recover bullion 950 fine, another will have 750 fine or less. This case is important, since it may be more expensive in the end, as purchasers of bullion pay for its assay value, and refiners' charges are more for base bullion than fine; under such conditions it may be profitable to use care. Bullion precipitated by zinc has objections which, although at times are in a measure unavoidable, can be circumvented to some extent by close application to details. The loss of bullion from smelting is one of the most objectionable features; the loss of zinc does not amount to much; the loss of cyanide is a bad feature, but unavoidable—we expect, however, to see

this reduced and the process carried on with much less cyanide in the future.

Practice makes perfect, and when the cost has reached as low as \$1 per ton the method is becoming nearly perfect. We also expect to see the recovery very much simplified, so that the recovered bullion will be finer and the process more readily accomplished.

The foundation having been laid, the house can be completed.

CHAPTER VIII.

THE RECOVERY BY THE CYANIDE PROCESS.

IF by amalgamation of a \$40 ore, 70 per cent has been recovered, it is considered excellent work; if now by the cyanide process and use of \$12 tailings 90 per cent more is recovered, we can consider it par excellent work, since we have recovered all but \$1.20 in the ore.

The Rand Extraction Company of South Africa treats tailings averaging 5 dwt. of gold to the ton equal by our figures to \$5 per ton. They recover 80 percent of this, or \$4 per ton. The writer has known from personal observation of the recovery of 98 per cent from pure quartz carrying gold, and of another instance of 97½ per cent from clayey ore, while on the other hand, for very good reasons, but 20 per cent of the fire-assay value was recovered. From this we deduce that the character of the ore has much to do with the percentage recovered, and moreover that the cyanide process is not applicable to every ore containing gold and silver.

If gold can be extracted economically without too great loss of quicksilver, amalgamation should precede

the cyanide process, for the cost will be very small, and we can almost surely count on the recovery of 90 per cent of the gold in the tailings by the cyanide process.

Again, as before mentioned, it requires longer time for cyanide to absorb coarse gold, than it does mercury. These remarks apply only to free-milling gold ores, and not to refractory or tailings ores, but it is in this last instance we appreciate the importance of the process most. Ores charged highly with metallic sulphides can be treated as before stated, but ores containing a high percentage of base metals cannot without first being oxidized. Ores carrying above 5 per cent metal are smelting ores; if they carry that much gold they are bonanzas: in either case if it be possible to send them to the smelters, unless free-milling gold ores, they should be.

The base and refractory ores generally contain such metals as iron, zinc, lead, and copper, besides such non-metal substances as antimony, bismuth, arsenic in the form of sulphides, arsenides, antimonides, etc.

Cyanide of potassium dissolves metals more or less, forming generally double salts. When the metals are not free, but are chemically combined with some other substance, such, for example, as sulphur, forming sulphides, they are not dissolved. It is well known that the affinity of cyanogen for the precious metals is more than for the base metals, but the weaker the solution of cyanogen, is the less likely are the base

metals to be attacked. The only sure method of determining whether the process is suitable for an ore is by laboratory tests. However, it is stated that should ores contain hydrated copper oxides, copper carbonates, or a considerable quantity of antimony, arsenic, etc., the cyanide process will not prove satisfactory.

Copper compounds physically hard are not acted upon, but when soft, porous, and open the action upon them decidedly interferes with the solution.

Dr. Scheidel found copper carbonate ore which while hard gave a very marked reaction on cyanide solutions. One fourth ounce of this ore shaken fifteen minutes with 2.73 per cent cyanide of potassium solution reduced the strength of the solution to 0.05 per cent cyanide. "The treatment of the ore in question, notwithstanding the rapid consumption of the cyanide, showed that 70 per cent of the gold was extracted from the ore, while no silver was extracted."

We would naturally expect that such ores would need preliminary treatment, and he gave it one with sulphuric acid, which he says "had a beneficial effect on the consumption of cyanide, and thereby on the extraction of gold."

MacArthur says "that cyanide solution acts very slightly on antimonial ores, but that these ores hold gold very firmly, and as the compound is impervious, the cyanide is unable to penetrate the mass and thus

dissolve the precious metal from the base." Both with copper and antimony cyanide solutions will act upon the gold, but with much of the bases present the preliminary treatment, together with the consumption of cyanide, will probably make the process unprofitable. This, however, would depend upon the value of the ore, as some ores can stand considerable expense and then pay.

An arsenical ore from the Mercur lode, Utah, carrying 20 per cent arsenic was treated at Denver. The results obtained were not such as to warrant the treatment of this ore by the cyanide process. On the other hand an arsenical ore from Boulder, Col. gave an extraction of 79.4 per cent, and still another gave an extraction of 82 per cent, at a treatment cost of \$2.50 per ton.

Why ores of the same class will not both yield readily to extraction has yet to be ascertained by the aid of the microscope and chemical tests.

The percentages of recovery depend upon so many things that are not understood that we can only say that the character of the ore is the chief obstacle to high recovery.

An average of eight concentrates from eight different mines gave a recovery of 88.13 per cent. The average quantity treated was 12.38 tons, and its average gold value per ton was 2 oz. 8 dwt. 11 grs.

From 43.5 tons of tailings, the assay value of which

was 1 oz. 11 dwt. 4 grs., 89 per cent of the gold was recovered.

From four samples of ore average weight 1.875 tons each, assaying 2 oz. 7 dwt. 5 grs. gold per ton, and carrying 17 oz. 9 dwt. 18 grs. silver per ton, the average recovered was 87.21 per cent gold and 62.19 per cent silver. Highest recovery of gold 91.09 per cent; lowest 81.18 per cent. Highest recovery of silver 79.78 per cent; lowest 35.16 per cent. The following table gives the recovery from different characters of ore, each from different mines—G. gold, S. silver:

	Per cent.	Per cent.
Ironstone and kaolin.....	G. 80.3	
Lead ore.....	G. 88.7	S. 59.58
Arsenical pyrites.....		G. 82
Hematite.....	G. 80	S. 61.55
Silver ore.....		S. 89.98
Iron pyrites	G. 98	S. 63
Stephanite.....	G. 90	S. 83.4
Siliceous ore.....	G. 94	S. 82
Mixed tailings.....		G. 80
Quartz and talc.....		G. 86
Talcose.....		G. 86
Fifty per cent sulphur pyrites....		G. 92
Oxidized ore with limonite.....		G. 93
Sulphuretted copper ore.....		G. 86
Sulphur, arsenic, antimony.....		G. 80

	Per cent.
Quartz and fine gold.....	G. 87
Rusty quartz.....	G. 93
Ore kaolin.	G. 100
Ore quartz.....	G. 100

The production as reported for five years from South African gold-fields shows the recovery on a grand scale. The process was applied mostly to tailings, and is here given in ounces: 1890, 286; 1891, 34,862; 1892, 178,688; 1893, 330,510; 1894, 635,-900. These figures are deserving of more than an ordinary glance, since they represent the progress made in one field since the introduction of the process in 1890; but especially are they deserving of consideration when they are known to represent ounces of gold discarded, thrown away as of no use, until the introduction of the process made their recovery possible. While it is not exactly in accordance with the ideas of the author to chronicle the cost of treatment, yet one comparison may not be amiss. Mr. G. S. Peyton of the Mercur gold mine, Utah, stated in print that while 1500 tons yielded 20 per cent of the assay value of his ore by amalgamation at a cost of \$4.25 per ton, 1600 tons treated with cyanide gave an extraction of 88.5 per cent at a cost of \$2.25 per ton.

CHAPTER IX.

LABORATORY-WORK.

HAVING followed the process from the stamps to the bullion, we can now turn our attention to the tests necessary to be made to intelligently conduct the work. Our first inquiry should be, How shall we ascertain the fitness of an ore for cyanide treatment? We must first crush our samples,—the finer the better,—and pass part of them through a sieve, keeping the remainder or coarser part for duplicate tests if they are needed.

To ascertain if the ore is acid and needs preliminary treatment, take 400 grams and shake it with water; if the water turns the litmus paper from blue to red, there is acid present. Add now lime, a little at a time, every now and then shaking until the litmus paper no longer turns red. The acid in the ore has become neutralized. Wash this ore thoroughly to remove any traces of lime. The ore is then ready for cyanide treatment.

There should be at least four tests made of 100 grams each of ore.

The ore is placed in glass bottles with solutions of various strengths.

These are shaken, some a longer time than others, the time varying, the object being to ascertain the amount of cyanide consumed by the ores in different times, and which strength is best suited for the standard or strong solution, as well as the time necessary for the reaction.

The cyanide consumed, the assay of the ore before treatment, and then again after treatment give the required data.

These tests will also determine whether an ore is suitable for treatment or not by examination of the amount of cyanide consumed. Percolation tests can now be made with solutions of various strengths and various lengths of time. Analyses now made of the percolations and assays of the well-washed residues will show the best treatment to use. These small tests are excellent guides for treatment on a larger scale.

Fire assays of ore to be treated should be made as the first step to ascertain the value or weight of the gold per ton; also that the process may be followed by assay to see how the cyanide is dissolving the gold. We must also measure our cyanide by this assay. We have seen that theoretically 1 part of cyanide will dissolve 1.5 parts of gold, but that practically we need from 3 to 4 pounds of cyanide for 1 ounce of gold; the amount actually necessary therefore can only be determined by the shaking and percolating

laboratory tests. One object is to be borne in mind while making these tests: that it is not how much, but how little, cyanide can be used and yet extract all the gold.

To determine the gold and silver in a solution of cyanide of potassium: Take a known quantity of the liquor and precipitate the silver with a solution of sodium sulphide or iron sulphide. Filter off the remaining liquor, dry, and wrap the precipitate in sheet-lead with a little granular lead and cupel. The remaining solution containing the gold is treated with chloride of zinc, which precipitates the gold. This is then dried, mixed with granular lead, and cupelled.

If there has been an excess of cyanide solution used, there will be a waste of cyanogen in treating the mass of ore.

To ascertain the strength of the cyanide solution: Take a burette and pour into it a solution of nitrate of silver, AgNO_3 , whose strength for 1 c.c. will equal the strength of 0.1 gram potassium cyanide used. Take a known quantity, 10, 20, or 50 c.c., of the cyanide solution being used, and drop into it the silver nitrate, shaking the cyanide to mix the two solutions. After a time the cyanide solution shows slight turbidity; when it shows distinct turbidity, stop, and read the burette. For example, if 20 c.c. cyanide solution titrated by 1.15 c.c. of the stand-

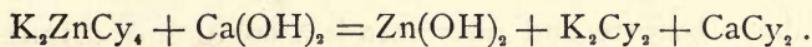
ard AgNO_3 solution, the strength of the cyanide solution would be found by dividing 1.15 by 20 = 0.057 per cent.

Mr. J. E. Clennell made some investigations which were published in the *Chemical News*, and again in the *Engineering and Mining Journal*. Extracts are given from his investigations:

1. Solutions of cyanide are frequently met containing such fine matter in suspension they cannot be filtered clear. If to such solutions lime and agitation be applied, they can be filtered clear.

When soluble double cyanides were present, this was inaccurate, but if only finely divided inert substance was in suspension it was practically correct.

With double salts such as are obtained from zinc filter-boxes the reactions made by lime are incomplete, hence valueless. Thus:



2. Turbid solutions free form zinc can be titrated without filtering by Fordo's and Gelis' method. The process depends upon the considerations:

a. That a mixture containing alkaline cyanides, hydrates, and monocarbonates has the hydrates converted into neutral salts, and the monocarbonates into bicarbonates, by addition of dilute mineral acid, before any of the cyanides are decomposed.

- b. That bicarbonates of the alkali metals do not act on iodine.
- c. When cyanide in such a mixture has been converted into a double silver salt, titration with dilute hydrochloric acid, with phenolphthalein as indicator, shows the quantity required to neutralize the hydrates and convert the carbonates into bicarbonates.

Iodine Process.—*a.* A solution of iodide of potassium is standardized against a solution of pure potassium cyanide, the strength of which has been determined by the silver nitrate method. The end point of the iodine reaction is marked by a permanent yellow tint, or if starch be added by a bluish-yellow color, and is in general sharp and delicate.

It is essential, however, that the solution shall contain no free caustic alkali or alkaline monocarbonate, as these bodies react with iodine, rendering the indications too high and indefinite.

b. Silver nitrate is added to a measured volume of the solution to be tested until a permanent turbidity results, or if the solution was turbid to commence with until a distinct increase is observed. A drop of phenolphthalein indicator is now added to the same liquid and titration is continued with N/10 HCl until the pink color disappears. Another measured portion of the original solution is now taken, and a trifle less hydrochloric acid than was necessary to neutralize

the alkali in the preceding experiment is added drop by drop with agitation. The solution is now ready for titration with iodine.

The following experiments illustrate neutralizing the alkali by hydrochloric acid:

Pure cyanide and caustic potash were mixed so that 10 c.c. of the mixture required 3.7 c.c. of standard iodine, and a like amount required 1.15 c.c. of standard silver nitrate.

On adding phenolphthalein and titrating with N/10 HCl acid 5.75 c.c. were required. 10 c.c. were then mixed with 5 c.c. of N/10 HCl and the mixture titrated with iodine; 3.3 c.c. were required.

Next 10 c.c. were mixed with 5 c.c. of N/10 HCl and the mixture titrated with standard AgNO₃; 1.15 c.c. were required.

In these experiments

$$1 \text{ c.c. standard } \text{AgNO}_3 = 0.01 \text{ gram KCy};$$

$$1 \text{ c.c. " iodine} = 0.0035.$$

To show that these are practically correct we will calculate them.

$$1\text{st. } 3.7 \text{ c.c.} \times 0.0035 = 0.1295 \text{ per cent.}$$

$$2\text{d. } 1.15 \times 0.01 = 0.1150 \text{ " "}$$

$$3\text{d. } 3.3 \times 0.0035 = 0.1155 \text{ " "}$$

$$4\text{th. } 1.15 \times 0.01 = 0.1150 \text{ " "}$$

The difference between one and three shows the

advantage of neutralizing the hydrate with acid, while the difference between two and four shows no variation when silver nitrate is used.

When an impure solution of cyanide and caustic were used, the advantage to be gained by the use of acid was in favor of iodine, a trifle.

A mixture of cyanide and carbonate gave with silver nitrate 0.044 KCy; with silver nitrate and acid the same result—0.044 KCy. With iodine the indicated strength was 0.047 KCy.

The next experiments were made with solutions containing zinc, but satisfactory results were not obtained.

3. *Precipitation by Akaline Sulphides.*—*a.* Zinc, silver, or mercury existing as double cyanides are precipitated by sulphuretted hydrogen or an alkaline sulphide, as metal sulphides.

b. The excess of sulphides may be removed from the cyanide solution without affecting the cyanide by addition of insoluble compounds of lead, such as oxides, carbonates, etc.

The method to be followed is:

First. Measure a volume to be tested; add caustic potash or soda until the cyanide solution is strongly alkaline.

Secondly. Pass into the now alkaline solution H_2S until precipitation ceases, or, what is better, a concentrated solution of pure sodium sulphide. Shake the

solution well and allow the precipitate to subside. The clear solution after filtering can be freed from excess of sulphide by agitation with litharge added little by little, until a drop of the liquor no longer gives the slightest coloration with a drop of lead acetate solution. A definite volume is now filtered off and tested with silver nitrate.

The liquid to be tested must give a perfectly white precipitate with a drop of lead acetate. Also it should give no precipitate with sodium carbonate, and moreover it should give no precipitate with sulphuretted hydrogen.

In titrating with silver nitrate the point to be noted is the appearance of a distinct permanent turbidity pervading the liquid and not disappearing on standing.

Deductions made were, that wherever a quantity of zinc was present it was necessary:

First. To remove the precipitation by filtration before adding lead salts, since the precipitated zinc sulphide, although washed, will react upon carbonate of lead with formation of lead sulphides;

Second. A solution of lead acetate is liable to precipitate cyanide of lead; hence an insoluble compound of lead should be added to the solution;

Third. To avoid the liberation of hydrocyanic acid considerable alkaline hydrate should be added before precipitating with hydrogen sulphide;

Fourth. Results obtained with alkaline monocarbonates were not satisfactory.

The following experiments will illustrate the application of the method:

Suppose we take 100 c.c. of the liquid from what has passed through the zinc-boxes, and which contains K_2Cy_4Zn .

We make this strongly alkaline by addition of concentrated caustic soda, and by addition of hydrogen sulphide precipitate sodium and zinc sulphide.

This liquor is then filtered, and, if necessary, a little lime is added to make the filtrate clear.

This filtrate is now shaken with an insoluble lead compound, such as litharge or lead carbonate, and 25 c.c. are now filtered off.

Supposing our pure potassium cyanide is 1 c.c. = 0.231 KCy, and that 1 c.c. of zinc sulphate = 0.02 gram Zn. Then in the double cyanide K_2ZnCy_4 , 1 c.c. = 0.08, and the free potassium cyanide = 0.231 - 0.08 = 0.151 KCy.

On titrating with $AgNO_3$ we find it requires from our burette readings 5.7 c.c. $AgNO_3$ for distinct turbidity.

$5.7 \div 25 = .228$ per cent KCy; but four times this gives total cyanide or 0.912 per cent KCy as the indicated strength of the solution.

GENERAL INFORMATION.

Briefly, the cyanide process, leaving out the mechanical arrangements for doing the work, is conducted as follows:

The ore after reaching the mill is crushed, rolled, and stamped. If amalgamation is practised, it takes place during the stamping. If wet crushing is employed, three to five gallons of water per minute will be required. The tailings from the stamps are conveyed directly into vats by means of launders.

The vats may be made of iron, coated with tar or asphalt; or of brick, lined with hydraulic cement; or of wood coated with asphalt or lined with cement.

Each vat, if the MacArthur-Forrest process is being used, should be large enough to hold one day's run from the stamp-mills. These vats are made of enormous size, at times fifty feet in diameter, capable of holding five hundred tons of ore and solution.

Wherever possible, the plant should be so arranged that advantage may be taken of gravity, as this will obviate the use of elevating machinery and much pumping. The leaching-tank, or the first vat of the series which holds the ore and solution, should have a capacity of twice the second tank of the series, which holds only the gold in solution.

The tank next in order is the precipitating-vat or, as it is termed, the zinc-box, and this should have

compartments enough to allow the precipitation of all the gold from the solution.

Following the precipitating-boxes are the sumps, which receive the weak liquor from them, and should be of such size as to hold the solution from two leachings.

There is still another tank called the stock-tank which contains a strong solution of potassium cyanide, and there should be yet another to which the weak liquor from the sump may be pumped and brought up to the standard or strong solution by additions from the stock-tank. The sump solutions are used over and over again by making up from the stock-tank the loss of cyanide which took place during the former operations, or they may be used as weak solutions at the commencement of the operation or as wash at the finish. In any case by using the sump liquor cyanide is saved.

The cost of cyanide 98 per cent pure is fifty cents per pound in New York City. It is costly and worth economizing. Since the process has increased the demand for cyanide of potassium, it is of importance to buy it from some firm which can be relied upon, and which tests it occasionally to ascertain its strength. The tendency, as the subject has been practically worked, is to greatly reduce the amount of cyanide used; where it was formerly customary to use 1½ and more per cent of cyanide in the solution, now

equally good results are obtained from the same ore by $1/2$ per cent for the strong solutions and $3/10$ per cent for the weak solutions.

Of course the ore and gold to be dissolved will have an influence upon the strength of the solution, also the silver, but the object of the operator should be to keep the solution as low as consistent with the work in hand, bearing in mind that the weaker the solution the less likely it will be to attack any base metals present, and the less loss will occur from oxidation and in the zinc-precipitation boxes. In this latter instance the greatest loss takes place, and for that reason inventors have endeavored to find some better method of precipitation. Some establishments instead of allowing the solution to go through the precipitation-boxes pump it back on to the ore or use it upon ore in another vat, thus making two or more treatments with the gold in cyanide solution before precipitating the gold by zinc, thus economizing in the use of cyanide by this means and saving zinc as well. This method, we are informed, is carried on at the Robinson works in South Africa, with extraction results equalling the former practice.

The numerous methods being devised for saving cyanide and lessening the cost of the process, together with the numerous patents being issued, must impress the thoughtful man, no matter how

prejudiced he may be, that there must be some merit in the process worthy consideration.

It is not claimed by any one we are aware of that all ores can be treated successfully, for such is not the case, but we believe that it can treat any ore where as much care is taken with oxidation as in other processes with equally good results and cheaper.

There may be given as one reason why better results have not been obtained in the treatment of tailings that amalgam on the surface of gold protects it to an enormous extent from the solvent action of potassium cyanide.

Another reason for bad results may be given, that not enough oxygen from the air was allowed to come in direct contact with the ore and solution.

The poor results obtained in the treatment of concentrates may be obviated by allowing more time for the cyanide solution to act upon them, or by finer pulverization. In both cases the action of the cyanide solution can be hastened by agitation.

In regard to the necessity of oxygen for hastening and carrying on the operation Mr. F. A. Mason deduced the following from some recent experiments:

First. That by passing air through solutions of potassium cyanide the solubility of gold was increased. This will be the subject of several more patents.

Second. That air when forced against a plate of gold in a cyanide solution causes it to dissolve with

more rapidity in that solution. This again proves the truth of Elsner's equation.

Third. That amalgam on the surface of gold, even when air is forced against it, protects the gold from the action of potassium cyanide. This experiment shows the weak action of cyanide upon mercury.

Ore-sampling.—The probabilities are that lean ores will afford the greater bulk of the ores treated by chemical processes, because they will not bear the transportation, sampling, freight, and other charges to the smelters.

In some instances, however, when the smelters may need "silica" for a flux so badly as to use pebbles, they will agree to stand one half the expenses, or even buy the ore; but such instances are likely to prove rare, as the smelters can afford such buying only in extreme cases, and even then only to a limited extent, depending upon the distance and freight charges from the mines to the smelters.

The smelters being generally situated in a locality suitable for obtaining different classes of ores, and nearer fluxes and coal on which they pay freight than the ore on which the miners pay freight, their usefulness for low-grade ores is a question, unless the concentrates of such ores are shipped them. Were the smelters compelled to treat all low-grade ores, there would be very few smelters, as then the actual running expenses would either bankrupt the smelters or

their charges would prohibit the miners shipping their ore. In any case the smelter is more useful for high-grade ores too rich for treatment at the mines, and for this purpose they are a necessity; while those ores, on the other hand, which can be treated at the mines by the chemical processes and yield good returns will not go to the furnaces. In both cases where the works are public, that is, buy the ore for treatment from the miners, the public sampling-mill is agent virtually for miner, and smelter, or reduction-works.

On the returns from the assays made by the sampling-mill the ore is paid for. The ore is not shipped direct to the smelter, but to the sampling-works. Here it is unloaded from the car and weighed. It is then run through crushers; from the crushers it passes through rolls; from the rolls it is elevated and comes down by gravity through a pipe which automatically samples it, by quartering it as it falls, and finally, after this mixing, receives one quarter in a separate bin, while the three quarters are removed to the dumping-bin.

After the car-load of ore has been sampled automatically in this way it would appear to be as accurately accomplished as any hand-work could possibly do it. But all the ore is not of the same value; hence the mixing of this sample must be attended to.

It is now removed from the bin and shovelled in a

cone-shaped pile on the floor, the ore always running off the end of the shovel to the apex of the cone, so as to run as evenly as may be down the sides of the cone. Throwing a shovelful of ore on the apex of the cone will not accomplish this result, but easy handling, as mentioned, will. This operation is done once or twice more, after which cutting down the sample is commenced.

To accomplish the cutting down the pile called the sample is shovelled from the floor on a sampling-shovel, which is made to catch about half of the ore from the floor, the remainder falling into a barrow. This sample thus reduced goes into three separate buckets, first one, then another, and finally the third, when the rotation is commenced over again. The original sample is now cut down one half, but is still very much too large, so that the ore from the three tubs is coned again on the floor as in the first place, and again cut down one half in quantity.

It is now run through the crushers and sifted, and again cut down, but before this cutting down all that did not pass through the sieve is crushed until it does. These three samples will now weigh about one hundred pounds each or less. This is too much, so they are still further mixed until reduced to ten pounds or less, when they are pulverized to pass through a 100-mesh screen. Then one part of each is sent to the assayer. Should there be much variation in

the assay returns, the operation of sampling is repeated when, if the assays tally closely, the average is taken as a basis for settlement.

It is well to remark that various ores contain more or less moisture, so that a lot of say 100 pounds is weighed, then dried, and weighed again. The total weight is now reduced just that much, for if a lot of 100 pounds contains 2 per cent moisture, a car-load of 10,000 pounds will contain 200 pounds, and the smelter objects to paying for that much water. After the assay has proved satisfactory to all concerned the sampler settles with the miner but first deducts his charges for sampling with the smelter's charges, and the sampler in turn settles with the smelter.

The sampling-works have become almost a necessity, as the miner feels better satisfied with his returns, and the smelter is satisfied because he is not accused of fraud. Whether the samplers are satisfied we are not sure, but presume they are, since public sampling is now carried on in every camp of size.

There are many cases where the miner cannot afford to loose the time and be to the expense of following his ore to the sampler; he therefore hires an agent who attends to the sampling. This man is called the miner's agent; he checks the weights, the moisture, and watches the sampling.

The sampler's scales are regularly tested by the pub-

lic scale inspector, so that every possible safeguard is afforded the miner against loss of values. He can, however, if he wishes sample his ore and have his assay made before shipping, but we consider it doubtful if he did not employ the sampler whether he could have settlements with the smelter on his returns if they differed much from the smelter's assay. The ore to be sampled at the mines is first broken by passing it through a crusher. From the crushed ore every fifth shovel is reserved for sample, the remainder being carried into the railroad car for shipment. This sample is now crushed finer by passing it through the rolls; it is then coned on the floor as above, flattened out, and divided into four parts, the two opposite being saved as the sample, while the remainder is taken into the car. This is again coned and quartered until the sample is much reduced in size, say to one hundred pounds. It is then weighed, afterwards heated to drive off the moisture, and crushed to go through a very fine sieve, after which it is reduced to size necessary for assay by the same method. This at times gives very accurate results, and is a check upon agent, sampler, and smelter.

CHAPTER X.

GOLD AND SILVER SOLVENTS COMBINED WITH ELECTRICAL ACTION.

ACCORDING to Berzelius, “Chemical union of any two substances is an electrical act; that during contact previous to union the one substance is relatively positive, the other relatively negative, and the act of union is a consequence of the attraction existing between the substances in these two states; also that in the act of uniting the two electrical conditions neutralize each other and produce heat.”

Professor Bunsen used to say that electrolysis in all operations marks a great advance on chemical action pure and simple.

As MacArthur and Forrest were the parties to introduce the practical application of cyanide to extraction of gold from ores, Siemens-Halske may be considered the first to introduce electricity into the process to assist its application. Neither of them designed or produced anything new, but, having strength in their convictions, they applied their patents to obtain useful results.

A good thing laying dormant is of no use to mankind.

When Dr. Siemens took the matter up, he found that electrical precipitation was equally effective with either strong or weak solutions of cyanide. This was one step in the right direction, namely, a saving in chemicals.

Julian Rae of Syracuse, N. Y., patented an apparatus as early as 1867 for the electrical precipitation of gold from ores, but the patent, like Simpson's, of 1885, seems to have been left on the shelf to mould. Mr. Rae did, however, make one practical test, which was not entirely satisfactory, at the Douglas mill, Nevada. His failure to obtain uniform results was due, we believe, to his using an alternating current. Electrolysis with such currents must necessarily be very slow, and the slower the reversal of the current the better the results, for if the current alternates quickly there will not be given time for the solution to deposit its metal compound before it is repelled by a reversal of the current.

The essential conditions necessary for electrolysis are that the substance be a liquid, a definite chemical liquid, and a conductor of electricity.

Potassium cyanide with gold in solution answers these conditions.

When potassium cyanide delivers up its gold to the zinc in the zinc-precipitation boxes, it would have its

potassium immediately converted into potash by oxygen did not potassium cyanide have a definite chemical composition of its own so that it could unite with zinc quickly. The zinc, being the substance liberated, unites with the potassium cyanide to form zinc potassium cyanide, and is a secondary product of electrolysis.

Not only precise quantities of current, but definite amounts of electrical energy, are required to decompose or separate weights of substances. The amount of electrical energy required to separate from a compound a given weight of metal depends upon the strength of the chemical union, the chemical equivalent of the substance to be separated, and upon the amount of the conduction resistance of the electrolyte.

The question of a cyanide solution being increased in dissolving power by becoming an electrolyte is not quite clear to the writer. That gold is positive there can be no doubt, and that cyanide of potassium is negative as an electrolyte there can be no doubt; but we are told that the solving action takes place at the anode; hence the gold positive dissolves to the cyanide solution negative, with the result that the gold cyanide solution is positive. The gold is able to keep its electro-positive condition if not its chemical condition.

Whether this electrolyte becomes converted into an electrode by absorbing the gold we are unable to say,

but when they become "cations" the gold is in the metallic state and the potassium cyanide is immediately set free. The potassium also should be set free, forming caustic potash, but if it be set free the majority of it reunites with hydrocyanic acid, forming potassium cyanide, to go on another round as a negative electrolyte to collect more gold. Of this much we are confident, but how the electric current stimulates the solution we are unable to comprehend.

That some potash is set free at the cathode we are sure, but the majority is reconverted into potassium cyanide.

As the chief amount of conduction resistance is in the electrolyte, and not in the electrodes, and by rise of temperature the resistance of the liquid is decreased, the effect is to facilitate electrolysis. At the same time this rise in temperature should increase the solubility of the solution.

The electro-chemical equivalents of gold and potassium cyanide are nearly equal. The equivalent weights are: gold 65.4, KCy 65.

The quantities in fractions of a gram separated by one coulomb of current are: gold .000678, KCy .000684. Grams separated per ampère per hour: gold 2.5128, KCy 2.4624. The amount of electrical energy, therefore, required to separate these substances being about equal for each on account of nearly equal atomic weights, we naturally conclude it

is small, and that an increase of current past a certain point would be a waste of power.

The object of the electric current, then, as we understand so far, is to decompose a solution of metallic salt, depositing the metal upon the cathode or negative pole, while the metal is dissolved at the positive pole.

If we use an iron anode with sulphuric acid, we find gold is electro-negative to iron, while if we use the same anode in potassium cyanide, gold is electro-positive to iron.

If electro-positive and electro-negative metals be connected, as in the case of our gold cyanide solution, they form a galvanic battery of their own, and, being nearly of the same electro-chemical relation, it approaches perfection. Were there any data to go by, we could almost say that this was the reason why potassium cyanide had such an affinity for gold.

We are not sure that these little galvanic batteries have any action whatever on the results of the other particles of gold and cyanide uniting, or that they are a means of conduction between the electrodes, since there are so few of them in solution, becoming gradually less as deposition at the cathode takes place.

In a fixed time a given electric current will deposit a certain amount of metal, which will vary for different metals in direct proportion to their chemical

equivalents. One coulomb or unit of quantity of electrical current will set free 0.000162 grams of hydrogen. Equal weights of different metals in dissolving will yield unequal amounts of current. If, however, the metals are considered in proportion to their different weights, a very different relation is seen. One gold coulomb is obtained by

$$\frac{0.000162 \times 196.2}{3} = 0.010611 \text{ grain of gold.}$$

One silver coulomb is obtained by

$$0.000162 \times 107.7 = 0.01725 \text{ gram of silver.}$$

One potassium cyanide coulomb would be .010530. The rate of deposition of these various substances, or rather their dissolution at the cathodes, would be in one hour by one ampère of current

$$0.010611 \times 3600 = 38.19 \text{ grains gold;}$$

$$0.01725 \times 3600 = 72.28 \quad " \quad \text{silver;}$$

$$0.01053 \times 3600 = 37.90 \quad " \quad \text{KCy.}$$

To deposit one ounce of the metals or cause their dissolution from an electrolytic condition with the above current would require as many ampère-hours as grains deposited by one ampère is contained times into 480 grains. We can facilitate matters by increase of ampères of current up to a certain point,

beyond which excess of quantity does not precipitate faster, but injures the electrodes and electrolyte. Laboratory tests are the only sure criterion for the quantity of current necessary, but these tests will be founded on the above facts, with others, which are also determined in the laboratory.

Knowing the quantity of solution in which a certain percentage of cyanide of potassium is placed, and then ascertaining the percentage of gold in that solution, we can calculate the necessary amount of current to deposit that metal uniformly. Every different electrolyte must be electrolyzed at a particular rate in order to continuously obtain from it the desired quantity of metal.

Increased density of the current is attended by increased heat of conduction resistance in the liquid, which has the effect of dissolving the anode, and in our case nothing more. "It is not by misdirected strength of current, however great, that reducible elements are separated." From a weak solution of a potassium salt even the strongest current will not enable us to obtain the metal; but by using a cathode of mercury of small surface the metal has been deposited by the aid of a feeble current. It thus becomes evident that the energy must be intelligently directed.

To liberate 480 grs. of gold it would require $4.80 \div 38.1 = 12.6$ ampère-hours, and to liberate

480 grs. of potassium cyanide from the solution would require 12.7 ampère-hours; but in our solution, following Elsner's equation, for every two parts of gold liberated we must liberate two parts of KCy (potassium cyanide), hence the number of ampère-hours would be $12.7 \times 2 = 25.4$ for potassium cyanide. Thus to liberate one ounce of gold from a solution of $2K_2AuCy_2$ would require 386 ampère-hours, or if the work was to be performed in 10 hrs. 3.86 ampères perhour for 10 hours.

By using more ampères we of course waste power, and injure the electrolyte for future use by oxygenizing it so that its potassium is lost, hence free hydrocyanic acid set free. *Cy*

CHAPTER XI.

CYANIDE SOLUTION WITH VARIOUS ELECTRODES.

ELECTRO-GILDING has been in use over half a century. Dr. Wright of Birmingham, England, used gold cyanide solution for electro-gilding in 1840.

The principle is that of depositing gold from a gold solution upon a cathode of metal to be plated.

The current or circuit is made so that electrical action must take place between the electrodes through an electrolytic compound of liquid. The current decomposes the liquid at the cathode, giving up its metal or liberating the electrolyte and its metal constituent from solution.

Elsner's equation has with electrical action the same features as it possessed where cyanide and zinc precipitation were employed. The oxygen decomposes the potassium of the potassium cyanide as before, except more briskly.

In this instance agitation is not as important for generating oxygen as it is for moving the ore and liquid so that the electrolyte can come in contact with the electrodes, in one instance receiving life for dissolving, in the other delivering up its metal and, becoming regenerated, and sent on its way to dissolve more gold.

By electrical action the cyanide can be greatly reduced in the liquor. As we have seen, theoretically 1 part of cyanide solution should dissolve 1.5 parts of gold, but that practically it requires 48 to 64 times that.

By the aid of electricity in connection with cyanide the amount needed has been reduced to 16 parts cyanide to 1 part gold, and we are of the opinion it will approach nearer the theoretical limit as matters are improved. As the solvent action takes place very much quicker for some cause, and as there is not the loss of cyanide as in percolation and zinc-precipitation, a very great saving in expense and time is recorded. Another item of consideration is that the liquid remaining will not contain the impurities deleterious to future action in anywhere near the proportions that it must in the former process. When we considered the former plan, we noticed that two or three changes of the solution were practised, with intervals between during which oxygen was allowed to act upon the ore, to furnish the necessary electro-negative element to unite the cyanide and gold.

By the use of electricity the electro-negative element is always present at the anode; this allows of the operation being continued without loss of time for percolation, and of its being continued, so long as any potassium cyanide is in the solution, or any of the precious metals to be dissolved. We have stated that agitation with the current passing is not so necessary

as in the former practice, nor would it be as far as oxidation and precipitation go, but there are other considerations which make it desirable that the sludge be kept in motion.

The solution is very weak, and in order to hasten its action mechanical power should be used. While it is possible, that by liberation of gases, and the action of gravity due to different densities of the electrolytes, either with or without gold in suspension would create circulation, still the fact is that the sludge would settle around and on one or more of the cathodes, thus hindering, if not entirely stopping the deposition of the metals on the cathode. Were the liquid clear or filtered from the sludge, this would not be the case, but by filtering we lose time and can not perform, neither can the cyanide do the work as readily as it is accomplished by agitation.

One object of using electricity is to save time, cyanide, and zinc; to do away with several tedious filterings or percolations, as they are called; and to do what leaching is required with one solution, at one time; and precipitate the gold at the same time that the operation is carried on in the vat. To accomplish this our apparatus, must with the solution satisfy the following conditions: There must be a metal anode not easily corroded or dissolved by the solution. The proper position of this anode is above the cathode; but it can be parallel to it if both are perpendicular.

When horizontal and in courses, the oxides from the anode fall away, and being deposited on the cathode will interfere with its action, especially if they can unite chemically with it. Secondly, the cathode must allow the gold to adhere to it, and after the process is completed allow the metal to be easily recovered. When the gold must be dissolved from the cathode by acid, such sheets of thin metal as can be readily removed from the vat should be used, the thin gold can then be removed by scraping the plates. These plates should also be arranged so as to be easily lifted out and returned to the vat, or if mercury is used it must be arranged to run off readily so that it can be strained and returned to the vat. Each inventor has the best mode of accomplishing these results, or at least did have until some one else overtopped him.

As power must be used for agitation, it must be such as will stir up the whole mass and keep it in motion. It occurs to us, the anode being the dissolving-point and the cathode the receiving-point, that if the power be applied to either one, that is, the mechanical power, it should be applied to the anode. If we can apply this power to either electrode, we have simplified matters one step more, by lessening mechanical complications.

As in the former process, the easiest and best way of recovering the precious metals is in the form of amalgam.

The objection offered by Von Gernet to a mercury cathode is impracticability, on account of the large amount necessary to carry on the operation. While we agree to this as far as the large amount goes, we must disagree with him on its being impracticable, considering it also economical as well to use mercury for cathodes. Von Gernet says that "sheets of copper coated with mercury have been used unsuccessfully, because the mercury, owing to the action of the current, will penetrate the copper, forming a dry amalgam which does not adhere to the plate." If Mr. Von Gernet will try to get that dry amalgam off from a copper plate, he will find that it does not fall off readily, and will not penetrate the copper more than during ordinary copper-plate amalgamation of the gold mills, but it gets against it some way decidedly tighter.

Of course the mercury will become dry in time, in other words, will wear out; to avoid this recover the gold once in a while, and put on a fresh coat of mercury. The cathode of lead which is used by Siemens and Halske has the disadvantage of wearing out, and requires to be as thin as is consistent with mechanical stability. It is used, however, to smelt the gold in recovery or refining of the deposit, and is a very long way ahead of zinc recovery in that respect. Siemens and Halske thus made two great improvements over the old process, viz., saving of cya-

nide and zinc, and the attendant troubles in refining.

It is not in our province to criticise any one; we wish to ascertain just what the improvements are which tend to make the subject more applicable to the extraction of gold from ores, and to point out defects as we see them. Should, therefore, an amalgamated copper plate be placed in a vat with the amalgam dry on it, the chances for its being scraped off by abrasion or grinding of the ore is possible, especially if the mercury be small in amount and the liquor circulating strong. The electrical current, together with the adhesion of the mercury to the copper plate, keep the gold in contact with the plate. If too much mercury be used, it will be forced into a free state, but if enough is used to keep the amalgam plastic and no more, it will not be rubbed off. However, to avoid such chances as these mercury in large quantities is poured into the vat to act as a cathode. The action of the liquid in circulating may give this liquid mercury the same movement as its own, but as it is denser than the liquid it only partially assumes the motion. Still it is sufficient, where horizontal circular motion is employed to move the mercury, to at times uncover the bottom of the vat in places. Just at this point mercury cathodes fail, for the molecule wishing to deposit its atom of gold may hit a bare spot and the next moment be lifted away. To overcome this

difficulty a copper amalgamated plate can be used in connection with the mercury, so that if the liquid mercury is disturbed the atom of gold may be deposited on the copper plate. Other cathodes besides mercury and lead have been tried, such as carbon, but of these we will speak in another chapter.

CHAPTER XII.

THE CURRENT.

THE quantity and the energy of current required for the electro-cyanide process of dissolving gold and silver from their ores are variously stated.

If in a fixed time a given electric current will deposit a certain quantity of metal, as we are told, it will be in direct proportion to their electro-chemical equivalents. "This law holds good only for solutions strong in metal; but with dilute solutions, as used in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, and consequently decomposition of the water takes place. For this purpose to make the efficiency of the precipitation as great as possible constant diffusion of the solution is requisite." (Von Gernet.)

If we have in our ore one ounce (480 grains) of gold and two ounces (960 grains) of silver, and we wish to precipitate this quantity in 10 hours, we will require

$$\frac{480}{38.1 \times 10} = 1.2 \text{ ampères of current for gold,}$$

and

$$\frac{960}{72.28 \times 10} = 1.32 \text{ ampères of current for silver;}$$

but we have the potassium as well to liberate, which for gold in solution will be theoretically 1 oz. gold = 2 oz. potassium; hence

$$\frac{960}{37.9 \times 10} = 2.54 \text{ ampères.}$$

The total ampères would then be for the three metals 5.12 ampères per hour, but as the ampères for one are sufficient for all three, the highest, 5.12, may be considered the quantity required. Should we have one hundred tons of such ore to be treated, our voltage must be considered. Suppose in treatment we used one half horse-power = 275 foot-pounds per second; as 1 watt = .7373 foot-pounds per second, we would use 372 watts, but a watt is the product of ampères by volts, hence $372 \div 51.2$ gives the voltage as 7.2.

In copper-refining the electrodes, that is, the anodes and cathodes, are given the same superficial surface as nearly as possible. The reason for this is to afford as much dissolving-surface as plating-surface that the result may be uniform. In our instance where the anode is stationary this may be followed, but where the movement of anode relative to the cathode takes place this is not possible. Again, where mer-

cury is the cathode, it is well to have it as small as possible, but not smaller than the anode. Von Gernet says "that a better effect is produced by doubling the surface of electrodes than by increasing the current tenfold." He was speaking of the Siemens-Halske precipitating-vat.

The iron anodes in that vat or series of vats had 14,000 square feet of surface, while the cathodes had 12,000 square feet of surface and were of lead.

The current of electricity used was from a 5 H.P. engine, and yielded $3\frac{1}{2}$ H.P. in electrical energy at 4 volts pressure.

$$\frac{746 \text{ watts} \times 3\frac{1}{2} \text{ H.P.}}{4 \text{ volts}} = 652.75 \text{ ampères.}$$

Anodes of iron, carbon and zinc were tried; the carbon could not stand the action of the current and it was abandoned; while zinc formed a white precipitate and was otherwise objectionable. Iron was finally adopted, although it precipitated Prussian blue. Iron as an anode is little acted upon by cyanide of potassium, but if the electrolyte is overcharged it acts upon the iron to corrode it. Wrought iron is more readily oxidized than cast iron when used as anodes. It requires a definite amount of current to oxidize a definite amount of iron, and in this instance 1080 lbs. were oxidized in a month; the gold precipitated in that time was 335,935 grs. in 700 hours, or 4709 grs. per hour.

According to the gold ampère-hour of 38.1 grs., it requires 8817 ampère-hours to deposit this amount. But Butters and Smart say 300 ampères were used, and that is to say 210,000 ampère-hours were expended or 201,283 more ampère-hours were expended than required. The current necessary to deposit this amount of gold was 12.6 ampères.

Hence the efficiency of the current $\frac{12.6}{300} = 4.2$ per cent, and the wasted energy was expended upon the iron.

As we stated before, the electrodes should be as nearly of a size as possible when we want to obtain equal results. They should be, especially the anode, of a metal or substance not easily acted on by the solution; iron seems to be suitable to fulfil the conditions of the anode, as it is not readily acted upon by the solution, and when oxidized it is due in great measure to having the current too energetic for the work in hand.

There must be a certain amount of energy, but not too much. The amount should be determined, as we have tried to show, by the work in hand to be performed. If to-day a two-ounce gold ore is to be treated in ten hours, more ampères will be required than if a one-ounce ore were to be treated to-morrow in the same time, and the oxidation of the anode will not be more, since surplus energy has not been expended upon it.

The question of the size of anodes and cathodes, that is, the surface exposed where agitation is used, especially iron and mercury, is purely theoretical. The anode in some cases is not more than one twentieth the surface of the cathode, and yet the iron anode is not very materially oxidized. This may be due to the movement of the anode, which rotates horizontally over the cathode of mercury, but at some distance from it. However, it does not seem possible that this rotation has much to do with it, since the current is continuous, and the voltmeters and ampèremeters do not show any variation of importance. We would suppose that energy would be expended more upon this anode, but we must take the facts as they are.

It has occurred to me the electrolyte was a good conductor of the current, and that not being often in contact in a state of metalloid with the cathodes, conducted the current, which was proportioned right for the work, to the anode and cathode, as if it were part of a circuit. While the heat arising from the conduction resistance throughout the electrolyte naturally reduces the resistance, it is small, and that leads us again to think that potassium cyanide with pulp is a good conductor. The quantity of energy lost by the corrosion of the cathode is very small, as the mercury does not corrode appreciably.

CHAPTER XIII.

ANODES.

THE anode, or positive pole, is the one by which the current enters the solution, and it is at this pole that the dissolving action takes place. Anodes not readily corroded by chemical action resist the passage of the current. For instance, an iron anode in a solution of potassic cyanide requires considerable electro-motive force to produce electrolysis. Little gas is evolved at the anode, and it is not corroded. Should gas evolve at the anode, the current passes readily, and the anode dissolves rapidly. With an anode composed of an alloy, the most easily corroded metal is first attacked. At times the anode does not dissolve, and becomes coated with an oxide, which hinders the current, but that anode which offers the least resistance to a current will dissolve most readily.

Usually the loss of the anode is greater than the gain of the cathode, but very frequently less than theory indicates, because while chemical action is tending to unite the constituents, the electrical action of the current is trying to separate them. We were told that when a non-corroded substance is used as anode an

elementary substance is set free at that electrode, but these elementary substances set free may unite chemically with other substances present and form new compounds. To avoid this in our case an anode should be used which will not form substances detrimental to the cyanide solution for future use.

Let us now consider some of the anodes used by the various inventors in their researches to obtain the gold and silver from ores by electrical action with cyanide potassium salts. Julio H. Rea (patent 1867) used as platinum anode. Platinum as an anode offers great resistance to the passage of an electric current; if silver is taken as 1, platinum offers 97 times more resistance. This quality is objectionable at the start, since it causes heat, and offers more resistance to passage of current, thus decreasing the conductivity of the electrolyte. It also takes, on account of resistance, more time or more electro-motive force to deposit the same amount of metal in a given time. Owing to the fact that decomposition must take place at the anode of the electrolyte, energy is expended, not on the work, but in setting free oxygen from the electrolyte, with the possibility of decomposing it. Another objection to platinum would be its great cost. Still in an ampère-hour the amount of dissolution of platinum should be, since it is a tetrad, 28.62 grs., or $\frac{3}{4}$ as much as gold. There do not seem to be others than Mr. Rae who suggest platinum as an anode, due,

no doubt, to its cost, which, for practical work, would be exceedingly large.

Carbon is suggested by Malloy's process, 1884 and 1886. In this case, as with platinum, oxygen escapes. The carbon is, however, practically unacted upon by the anion chemically, but the current or the oxygen mechanically disintegrates the carbon. The greater the current the more rapid the disintegration.

It is further objectionable because of energy of current lost, and possibly the active energy of the electrolyte.

Eissler in speaking of the Halske-Siemens process says: "Carbon could be used as an anode, but will not stand the action of the current, and soon crumbles into a fine powder which decomposes cyanide." "This finely divided carbon is in suspension, and cannot be removed from the solution by filtration." On the other hand, Mr. Weightman used arc-light carbons for six months and they were still in good condition. Mr. Weightman's observations do not agree with Mr. Eissler's, but that is not due to error on either side. One undoubtedly had reference to porous carbons, while the other had reference to dense compact carbons, coated with copper. The copper coating was of very great assistance even to the compact carbon in counteracting disintegration of the carbon by the current. The great objection to carbon is its lack of conductivity; if silver be 100, then carbon as

graphite is .069, and as gas coke .038. An advantage in its use is its small power of dissolution, which is but 1.74 ampère-hours; hence it could cause little deposition without great electrical energy.

Cassel, H. R., used carbon rods as anodes, as did Fischer and Weber (patents 1883, 1885, and 1887). Mr. J. B. Hanney used a mixture of plumbago and resin as anode, and, as he claims, with very good effect. As a rule, the anodes have been varied but little, but in some patents they have had different arrangements, so as not to conflict.

Zinc as anode has serious defects, in fact as numerous as when used as a cathode. It forms a precipitate of cyanide of zinc in the solution; also zinc oxide on its surface, which offers resistance to the electro-motive force, making it expensive. Should it precipitate on the cathode, it will also cause future annoyance, and as one of the advantages claimed for electrolysis is that it does away with zinc impurities, should we consider zinc at all in connection with electricity, we have gone backward instead of forward in the perfection of the system.

Zinc will cause the solution and deposition of 18.95 grains in ampère-hour; hence it is only slightly better than iron as a conductor, while very much inferior as far as by-products from the operation are concerned. Barker's process, 1882, had for anode brass, and did not use cyanide solution, but had for its object the

preventing of mercury from sickening. Leaving out the anode and substituting cyanide solution for water, we believe his invention would be successful in a large degree; otherwise we would not mention it here. The subject of anodes for copper-refining and electro-plating is well known; the conditions of such anodes compared with anodes for the electro-cyanide process are so vastly different that we can only approach them in practice.

For copper-refining the anode is stationary, and the same may be said for electro-plating in general, but in our case we wish, if possible, to have movable anodes, which do mechanical and electrical work at the same time. Again the ever-changing classes of ore as far as the quantities of gold and silver in the ore are concerned, make the matter of proportioning the size of cathodes very difficult to meet theoretical requirements.

CHAPTER XIV.

CATHODES.

THE cathodes next require our attention, and these are as important as the anodes, and possibly more so. Upon these the metal from the solution is to be deposited, and as we wish to obtain that metal in such form as to refine it with least complications and skill, the most satisfactory cathode must fulfil conditions more trying than the anode. It is the pole of the current from which the current passes out of the liquid, so that any chemical decomposition or disintegration of the molecules of the electrolyte must take place here. It must be a good conductor of electricity and must not be readily corroded. As a rule, insoluble coatings are less frequent than at the anode, happening generally when highly oxidizable metal is separated. A film of hydrogen sometimes adheres to the cathode, which will slightly diminish the current.

The density of the current at the cathode needs to be so regulated as to allow the metals to be deposited as metals, and not as powder, and as we are dealing with compound substances, the cathode should be of such a nature as not to unite readily with the secondary products of electrolysis. From an impure solution

the least electro-positive metal should be deposited upon the cathode first; consequently with our solution containing gold and potassium cyanide, silver and potassium cyanide, as double salts of cyanide, the rate of deposition would be silver, gold, potassium; but it is doubtful under these conditions whether it is necessary to use current more than sufficient to liberate the gold and silver, allowing the KCy to remain in solution undisturbed. If this can be accomplished, there will be considerable voltage saved as well as cyanogen.

Unable to obtain any reliable information regarding electrolysis of mixed solutions, we can only draw conclusions from observations. We know, however, that the current will pass through an electrolyte when feeble so as not to cause electrolysis, and that aqueous solutions of the zinc and copper cyanides will dissolve and deposit equally on passage of suitable current; so we are led to infer from these phenomena, since the electro-motive forces of zinc and copper are equal, that the least abundant metal will deposit least; but if equally solvent in the solution the least electro-positive will deposit first, and this latter has been proven in the case of silver salt in copper solution. The size of the cathode should be larger than the anode if its conductivity is less, but may be smaller on the reverse of this proposition. It should be of sufficient size, however, to be able to absorb or receive as much metal as is offered for deposition.

If the vat is used with agitation, the cathode should be as large as the diameter of the vat, so that every chance for deposition be accorded the solution, and not be contracted by a small cathode, which must focus the current from the electrolyte.

Julio Rea used copper cathode without amalgamating it; Mr. Simpson used a zinc cathode; Michel Body (patent 1894) uses mercury in a contrivance; Mr. Hanney used mercury; Mr. Whitall an iron cathode; Chas. Raleigh employed copper saturated with mercury; Mr. Keith uses copper plates preliminarily treated with mercury; Siemens-Halske use thin sheets of lead; Barker used mercury, as stated; Fisher and Weber mercury and metal plates.

The employment of mercury seems to be the more general for cathodes than any other metal except amalgamated copper plates. Copper makes a good cathode, but as the precious metals are deposited upon its surface dry and hard, there are objections to its use without amalgam. The copper plate is very little wasted by the current. Mr. Simpson's zinc cathode is objectionable for reasons stated. Mr. Whitall's iron cathode was objectionable because the precious metals were deposited as powder, hence no agitation could be used; and is here further objectionable where electricity is concerned on account of impurities in the bullion.

Chas. Raleigh uses mercury, but his plan of pump-

ing mercury through between two active surfaces, as he says, makes the matter dense. (See English patent 14,910, 1893.)

Mr. Keith's mercury and copper plates will answer admirably, but he follows on the lines of Siemens-Halske, with the bad plan of using porous cells, and circulating the solution after leaching through boxes for precipitation. However, it does away with zinc becoming mixed with the precipitate.

Lead cathodes used by Siemens-Halske are very thin sheets of lead, and to obtain the gold from them they are smelted with the gold. The process has worked very successfully, but requires smelting skill.

Pelatan-Clerici process uses both an amalgamated plate and mercury; also carbon cathodes electrically copper-plated, then coated with mercury.

There are other processes which have peculiarities which we will touch upon in general, using the patentees' words, for we have had no chance to prove their claims. There are other electrical processes for different solutions than cyanide, which we do not attempt to follow.

Wm. Crookes (patent No. 462,535, 1891).

He says: "In carrying out my combined process I take the gold ore, tailings, etc., and mix with them a solution of nitrate or cyanide of mercury, and pass a rapidly alternating current of electricity through the mass, either when at rest or agitated in any manner."

"The bulk of the mass is not a good conductor of electricity, while the fine particles of gold are excellent conductors." "Iron or carbon can be used as electrodes, and each electrode is alternately anode and cathode." "Assuming that sulphate of mercury is the mercurial salt, the current liberates sulphuric acid at one pole and mercury at the other." "The action now being reversed, the mercury liberated previously has a molecule of acid to unite with it, so that at each pole the mercurial salt is decomposed only to unite again." Since the gold in the wet mass is a better conductor than the surrounding mass, the equipotential lines of force will converge toward them, so that more of the current passes through them than the rest of the mass, and the two sides of each particle of gold act as anode and cathode. On one side sulphuric acid is liberated, on the other mercury, but the affinity of gold for mercury is so great they instantly amalgamate on one side, finally on the other side. Thus the size of the particles is not essential, as the finest flour and float gold will be amalgamated. Nor does it matter to what degree of coarseness the ore is crushed so long as the mercurial salt penetrates to one part of the piece of gold locked up in the ore, for the action will then take place and the metal become amalgamated."

"The advantage incidental to the use of an alternating current is that the sudden and violent decom-

positions and recompositions cause the mass to become hot, and so greatly facilitate the amalgamation."

"The efficient action is dependent upon several variable factors, viz., current density, area of electrodes, rate of alternation per second, and electric conductivity of the crushed ore and liquid."

We are not aware that this process has had practical application. It may be successful in the laboratory on rich ores, but we have never heard of it beyond the patent specification. We have quoted it because cyanide can be used in connection with it, and it is one process which uses the alternating current.

CHAPTER XV.

VARIOUS PROCESSES AND CONCLUSIONS.

MR. MACARTHUR said that when cyanide is used in combination with electric current not only is there a larger expenditure of chemicals, but the base metals are dissolved to a large extent along with the gold and silver, and for subsequent separation involve extra expense, which is saved by our process.

We believe he was afraid of the Siemens-Halske process at the time he said that. However, we must take him at his word.

Mr. Hanney, a countryman of his, says that when the electric current is used the cyanide solution may be weaker, as its action is increased by the current. He further states that only metal and silver salts are attacked by the solution, and that copper and iron pyrites may be stripped of their gold and not be attacked otherwise.

He uses mercury for a cathode and obtains the gold direct from the ore, which is the easiest method of collecting it for subsequent separation. Another advantage he obtains in this collecting of the precious metals is that no previous treatment, such as amalgamation, is required for the coarser particles of gold.

The advantages claimed by Mr. Hanney are so broad that they almost amount to his having an ideal apparatus.

They are: That the precious metals are extracted direct from the ore.

That economy is practised in the use of cyanide for extraction, thus approaching more theoretically to the quantity of cyanide necessary for the dissolving of gold.

That the precipitated metals are at once obtained from the amalgam in the metallic state, thus requiring no chemical or intricate treatment after precipitation.

That any workable quantity may be treated, so as to dissolve and precipitate the precious metal, and recover the bullion in one day.

That when the process is completed the liquid from the tank may be run off through a rough filter to clear the liquid of sludge so that it may be used over again by being brought up to its standard for recovery.

Percentages recovered from three different ores by this process gave 76.2, 86.9, 95.3, an average of 86.1 per cent.

He further hastened the solvent action of the cyanide solution by agitating the solution by means of a screw propeller placed in the bottom of the vessel, which gave the liquor an upward movement along the sides, then down through the centre of the vessel so

as to come in contact with his anode of plumbago and resin.

His planning to obtain all the good points was admirable. Rae's was not quite as good, and Mr. Simpson did not apparently try to improve or push his invention. Siemens and Halske, and Dr. Keith's inventions were only broad enough to deposit the metals after percolation to avoid the trouble incident to zinc precipitation, and what advantage Dr. Keith gained by amalgamated copper plates he lost by using anodes in porous cups 1 inch diameter, 24 inches long, the cost of which would in all probability equal the cost of the lead by Siemens-Halske's process.

The most recent process before the public is the Pelatan-Clerici. They put forward the same claims as does Hanney; they say that while the simple cyanide process is not capable of universal application, their process is more applicable.

That where the recovery of gold by amalgamation is from 55 to 60 per cent they are able to recover 85 per cent, and more. They say that under the most favorable conditions the common cyanide process saves 68 per cent of the gold, and barely 50 per cent of the silver.

Their claims are that they use a more efficient solution and entirely different method of treating the ore.

The treatment consists of a single operation which

is carried on in a vat. This vat, in the centre of which is a stirrer of metal (iron) and which is composed at the bottom of special amalgamated copper plates, is filled with crushed ore or tailings.

At the same time a diluted solution containing cyanide of potassium, common salt, and some other chemical is added, the portion of each reagent being calculated according to the quality of the material treated.

The stirrer is then set in motion so as to mix the ore with the solution, and to make a liquid sludge through which an electric current generated from a dynamo is allowed to pass freely. The particles of precious metals are thus brought again and again into contact with quantities of cyanide much more than sufficient to dissolve them entirely.

Owing to the very efficient dissolving power of the solution on the one hand, and to the action of the electric current on the other hand, the fine gold and silver are readily dissolved, and afterwards electrically deposited on the amalgamated plates in the vat, where they can be recovered in the form of amalgam.

The coarse grains are amalgamated, so that they are also recovered.

This process admits of a far smaller quantity of cyanide being used than in any leaching process known.

The time required for working a charge is from six

to twelve hours, while by the ordinary method as many days may be consumed—under the most favorable conditions three days.

This process avoids all the difficulties and complications connected with the leaching and washing of the tailings, the recovery of gold and silver in solutions, and the presence of baser metals. The liquor, *i.e.*, solution and ore, is run out through an opening in the bottom of the vat, and the vat is then ready for another charge of ore. The waste liquor is allowed to settle in a tank below the vat, and after a time two thirds of it are drawn off to be used again. The liquor contains from .001 to .002 per cent of cyanide; it holds in the solution but faint traces of the base metals.

This process is said to work better with slimes than coarser ore.

The cost of crushing must be added to the cost of treatment of ore, but as this varies from 50 cents to \$2.50 per ton, depending upon circumstances, they do not consider it in making up the cost of the process. This crushing cost must not be considered in comparing the cost with tailings treatment which is the result of other treatment, unless the cost of crushing, amalgamating, and tailings treatment be combined, since those operations by the ordinary processes only accomplish as much as this one process.

The chemicals used are 2 to 3 lbs. potassium cyanide per ton of ore or 0.001 to 0.0015 per cent solu-

tion; besides sodium oxide (Na_2O) as an oxidizing agent, or lime. This reagent is not given, but its cost is placed at 50 cents per lb. As cyanide potassium is 50 cents per lb. and two thirds are recovered, the total cost of chemicals is \$1; labor and general expenses are placed at 80 cents per ton, 10 per cent interest and depreciation on plant placed at 20 cents per ton. Total cost of treatment \$2 per ton. The capabilities of this process are as wide as by the older system, with the advantage of recovery of more silver.

Conclusions applying to both methods, simple and electrical:

1. The plant required is comparatively inexpensive.
2. The extraction is arrived at without any previous treatment of ores, except ordinary crushing, which must take place for every treatment.
3. The extraction is simple, and quite complete; tailings can be treated successfully when failure has attended other processes.
4. The precious metals can be precipitated from the solution in various ways to suit the ideas of the operator, but the simplest way is by the electrical current with mercury cathode.
5. The simple cyanide process extracts only the fine gold, thus necessitating previous amalgamation with free-milling, and longer contact with the solution, and very fine crushing with refractory ore. The electrical

method with mercury requires no previous amalgamation.

6. The cost of treatment is not so high as to deter the use of either process, and at times is quite reasonable, depending upon circumstances.

7. The factor time enters into competition with machinery on the one hand, while machinery enters into competition with time on the other. We have neglected previously to state an important factor in favor of agitation, viz., that the tanks can be cleared more readily for another operation; with percolation they must be shovelled out.

8. Talcose and clayey ores, such as make leaching difficult even when mixed with sand, are readily treated by agitation and the electric process.

9. Failure to obtain satisfactory results has occurred from neglect on the part of the operator to properly oxidize the ore before applying the cyanide solution.

Failure has occurred in other instances because sufficient knowledge of the process was not possessed by the operator. Early experiments proved very unsatisfactory especially in the South Atlantic States, where this process seems to have been tested before it had attained its present state of perfection, or rather before the intelligence now possessed upon the subject had been acquired by experiment.

Since the early experiments were inaugurated the

process has gradually obtained prominence, not so much from writings as from actual results, until now we have come to realize its usefulness in the treatment of ores which we considered formerly of trifling value, and in some instances of no value whatever. Whenever failures are to be recorded in the future of this process, they will be due to the following: first, attempting to work ores not suitable for the cyanide process; and secondly, to the ignorance of the operator. Those who intend to erect extraction works in this enlightened age of mining should acquaint themselves with the process, and ascertain if it be applicable to the ores they propose to treat. It is not a difficult matter now to ascertain the fitness of ore for cyanide treatment, and after it has once been determined to be suitable for cyanide treatment, nothing but an absolute change in the character of the ore will interfere with its successful work, except, of course, the ignorance of the operator.

Mr. F. A. Moson's experiment shows that where tailings from amalgamating mills are treated, and the gold particles are coated with mercury, cyanide will not act upon them; this will account in some measure for not obtaining at times a higher percentage of extraction with the older cyanide method. This objection is removed by electrical cyanide process.

In the treatment of concentrates care must be used to have them finely pulverized for cyanide treatment

when they do not yield their gold readily by laboratory test.

The writer is indebted to the following people and publications for information and data (patents U. S. unless otherwise stated):

- | | |
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| H. W. Faucett..... | No. 236,424, Jan. 11, 1881 |
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The various writers whose names appear have written for the above publications, and to those interested are probably well known.

USEFUL INFORMATION.

Area of circle = diameter squared \times .7854.

Circumference of circle = diameter \times 3.1416.

Cubic inches in gallon imperial = 277.276.

" " " " U. S. = 231.

" " " foot = 1728.

Cubic foot = 6.232 gallons.

Cubic yard = 168.264 "

Imperial gallon weighs 10 pounds.

U. S. " " 8.33 pounds.

One ton water = 200 imp. gallons.

" " " = 240 U. S. "

One cubic foot water weighs 62.5 pounds.

Drachm avoirdupois = 27.3 grains.

" fluid water = 54.7 "

One cubic inch water = 252.5 "

Pennyweight Troy = 24 "

Ounce avoirdupois = 437.5 "

" Troy = 480 "

Pound " = 5760 "

" avoirdupois = 7000 "

Gram = 15.432 "

Grain = .0648 gram.

Ounce = 28.3495 "

1 coulomb = unit of quantity of electric current.

1 ampère = " " flow = 1 coulomb per sec.

1 ohm = " " conduction resistance.

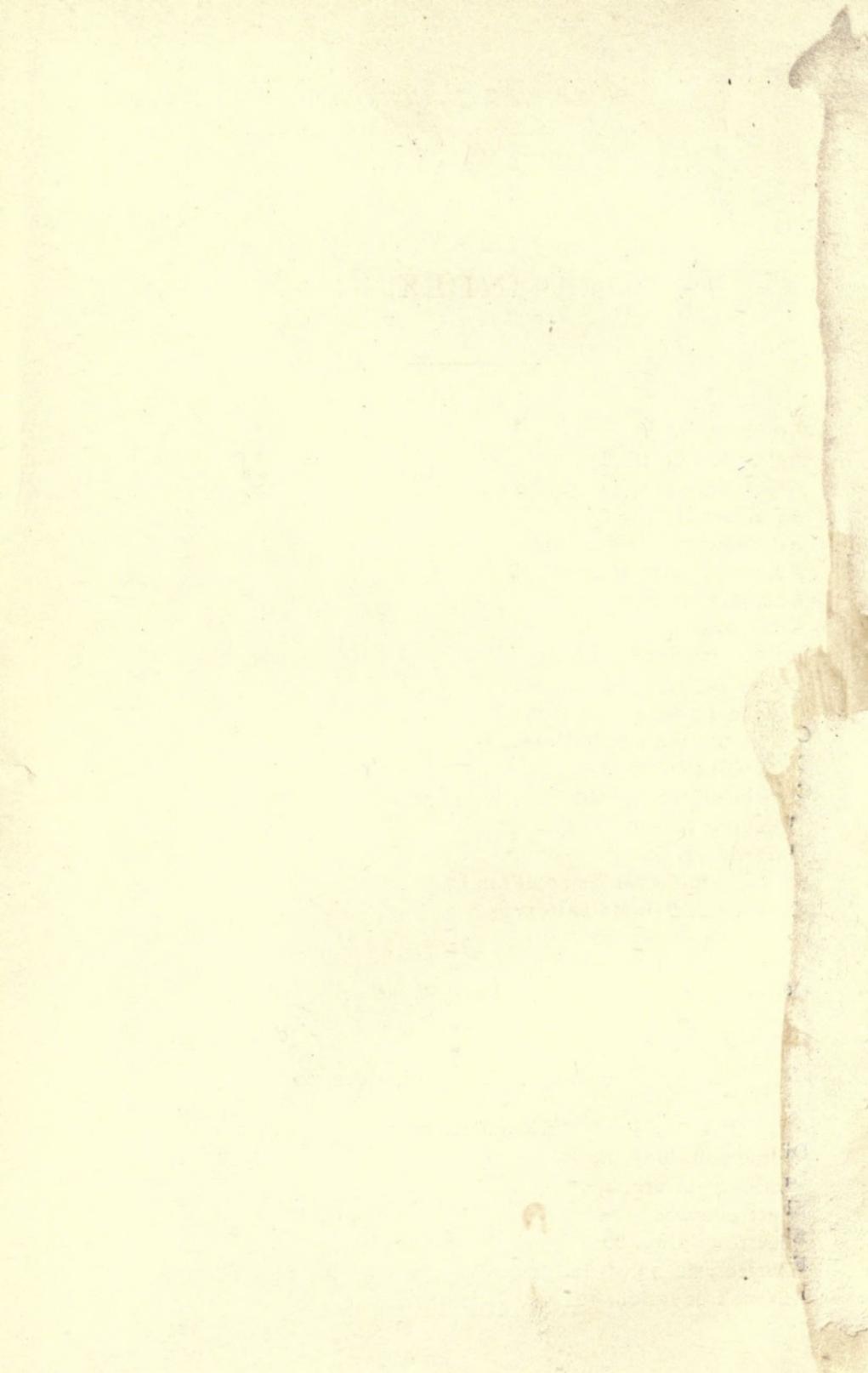
1 volt = " " electro-motive force.

Assay ton = 450 grains Troy weight.

" " = 29,166 milligrams.

Avoirdupois ton = 2000 pounds or 32,000 ounces.

" " = 29,166 Troy ounces.



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